MINING engineering

NOVEMBER 1956

WILFLEY
SAND PUMPS

INCREASED OUTPUT 1½ TPH!

The team of a Wilfley Sand Pump and a cyclone classifier increased this sand plant's production by 1½ tons per hour! In addition, the American Society of Testing Materials' spec. was raised approximately 50%. Because of the recovery of fines which were being washed away, and the dependable, trouble-free operation of the Wilfley Sand Pump, this producer—using the same quantity of raw materials—was able to upgrade the quality of products and increase profits.

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WILFLEY ACID PUMPS
"COMPANIONS IN ECONOMICAL OPERATION"
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MINING BRANCH AIME

MINING engineering

VOL. 8 NO. 11 NOVEMBER 1956

COVER

Increased conveyor use is symbolized by belt depicted on cover by artist Herb McClure. For the story of high speed photography, a new tool to reveal sources of conveyor belt wear and tear, turn to page 1089.

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PERSONNEL

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc. (Agency) operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Manager, B.S., 51. Thirty years experience surface and underground miner, engineer, foreman, superintendent, and manager, with record of profitable operation as well as increasing ore reserves. Worked in iron, copper, gold, lead, zinc, silver, tin, platinum, and tungsten. Prefer West or foreign. M-287.

Geologist, B.S., 31. Seven years field and mine experience. Free to travel. Desires position in mining or geology. Exploration, development, or mine staff. Aggressive, expanding company. U. S. or foreign. M-288.

Graduate Mining Engineer, 29, single. Nine years experience hard rock mining in Europe and Africa; desires position with mining concern, South America preferred. Available after agreement. Presently located Africa. M-290.

Mining Engineer-Placer Dredge Engineer, B.S. mining engineering, 36. Experience as chief mining engineer, large placer dredging corporation; 14 years varied experience metallic ore drilling, prospecting and operating, purchasing, personnel management, and field construction. Available reasonable notice. Prefer U. S., Alaska, and some foreign. M-289-SF.

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are regularly purchased by Macro Tungsten Refinery. For our quotation F.O.B. shipping point, write P.O. Drawer 440, Port Coquitlam, B.C., Canada.

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If requested, will be mailed on approval.

Mining and petroleum engineering graduate, preferably with practical experience, to instruct petroleum and mining engineering subjects and assist in mineral dressing. Opportunity for graduate work. Rank and salary open. Location: Dept. of Mining and Metallurgy, University of Wisconsin, Madison.

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EXTRACTIVE METALLURGY

To work on process development for new orebodies and process improvements for established plants.
Diversified challenging projects.
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Ore dressing and extractive metallurgy. Salary depends upon experience. Write to: Chairman, Department of Mining and Metallurgy, Texas Western College, El Paso, Texas.

POSITION OPEN

Tax Accountant as assistant to manager of Tax Department of a growing independent oil and mining company on West Coast. Prefer age 35 to 40 with considerable tax experience. Offers opportunity in tax department and other corporate obligations. Submit details of personal status, education, experience and idea of starting salary. All replies treated confidential.

Box 16-K AIME
29 West 39th St., New York 18

THE RIO TINTO GROUP

The Senior Mining Engineer and Technical Executive for the Rio Tinto Group will be retiring next year.

The Group are seeking as his successor a Senior Mining Engineer of the highest qualifications and standing in the profession, to advise on all technical matters and to coordinate the technical work of the Group.

Candidates should preferably be between 40 and 50 years of age and certainly not more than 55. Applications should be sent to the Secretary, Rio Tinto Management Services (U.K.) Limited, Barrington House, Love Lane, London, E.C.2., in time to be delivered not later than 30th November, 1956.

POSITIONS OPEN

Mining Engineers. a) *Mine Superintendent,* 35 to 45, mining graduate, with at least five years supervisory underground metal mining experience. Salary, $12,000 to $15,000 a year. b) *Assayer,* young, for analysis of complex gold, silver, lead, zinc, and copper ores. Salary, $4500 to $5400 a year. c) *Junior Geologist,* for mine mapping, exploration, and development work. Salary, $4800 a year. Location, West. W4061.

Mine Personnel. a) *Assistant Mine Manager,* with administrative, supervisory, and labor negotiatory experience required; experience with hard rock and trackless mining highly desirable. Salary open. b) *Mill Superintendent,* graduate engineer, with at least ten years experience in hydrometallurgy. Salary, $10,000 a year. c) *Metallurgist,* graduate, with at least five years applicable experience in hydrometallurgy. Salary, $8000 a year. d) *Chief Chemist,* graduate, for analytical and assay laboratory with some administrative experience. Salary, $6600 to $8000 a year depending upon experience and qualifications. e) *Chief Engineer,* adequately qualified and experienced to direct and supervise all underground and surface engineering and surveying. f) *Chief Surveyor,* experienced in underground

(Continued on page 1054)

RHOANGLO MINE SERVICES LIMITED
requires a
MINERALOGIST

for employment, under the direction of a senior mineralogist, in a modern research laboratory in Northern Rhodesia specialising in problems covering a wide field in the extractive metallurgy of base metals. Applicants (male or female) must have experience and interest in determinative mineralogy. Basic starting salary dependent on qualifications and experience with minimum of £978 per annum plus variable cost of living allowance (at present £62. 8. 0. per annum) and variable metal bonus (at present between 60% and 70% of basic salary). Leave 41 to 48 days per annum, accumulative up to 3 years. Generous pension, life insurance, and medical schemes. Accommodation provided at nominal rental. Replies, stating age, marital status, qualifications, experience record and availability, together with names of two references, and a recent photograph, should be addressed to:—

The Secretaries,
Rhoanglo Mine Services Limited,
P.O. Box 172,
Kitwe, Northern Rhodesia.

NEW "KING-SIZE" CARTRIDGES SAVE TIME AND LABOR

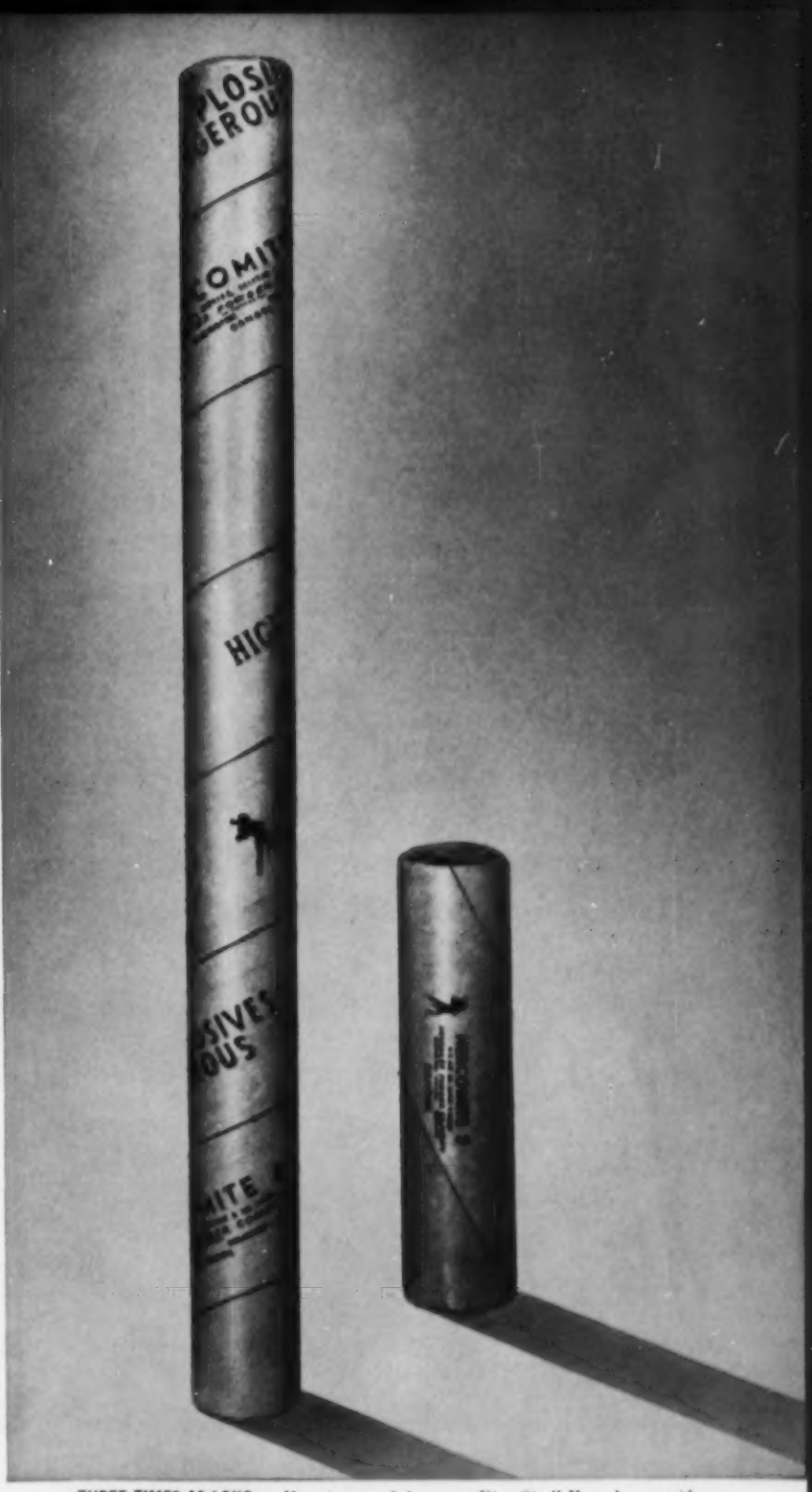
Now, Hercules produces "King-Size" cartridges in lengths of 24, 20, 16 and 12 inches, and in diameters of 1¼, 1½, 1¾, and 2 inches.

Available in all types of Hercules® dynamite, these long-length, small-diameter cartridges are now in full production.

"King-Size" cartridges make possible a more uniform fragmentation through the discharge of a single column of explosive. Their use means substantial savings in valuable time and labor in loading holes.

Hercules designed, tested, and installed special new packing machines for these "King-Size" cartridges.

Our technical service and sales representatives will be glad to discuss with you how these "King-Size" cartridges can go to work for you.



THREE TIMES AS LONG ... Here is one of the new "King-Size" Hercules cartridges—24 inches in length, shown alongside the same grade in the conventional 8 inch size.

HERCULES

HERCULES POWDER COMPANY
INCORPORATED
Explosives Department, 955 King St., Wilmington, Del.

Birmingham, Ala.; Chicago, Ill.; Duluth, Minn.; Hazleton, Pa.; Joplin, Mo.; Los Angeles, Cal.; New York, N.Y.; Pittsburgh, Pa.; Salt Lake City, Utah; San Francisco, Cal.

• Largest Pulverizer in

. . . Doubles Production for Pierce, Florida Plant

To meet a large increase in demand for ground phosphate, The American Agricultural Chemical Co. found it necessary to double the grinding facilities of their Pierce, Fla. operations. After a careful study of available systems, the Kennedy Air Swept Ball Mill System was chosen as the one best suited to grind pebble, concentrate, or a combination of both at the lowest cost. Dependability, continuity of operation, low power, maintenance, and operating labor costs were the prime factors in their decision.

The ball mill has long been used in many industries where continuous production with minimum maintenance and operating supervision is required to keep production costs to an absolute minimum. The simplicity of the ball mill, both in design and operation, makes it the ideal pulverizer.

The Kennedy-Van Saun Air Swept Ball Mill Grinding System is the ultimate in Ball Mill Systems. The Kennedy Radial Flow classifier makes possible production of a wide range of product sizes, by positive external adjustments, to a high degree of accuracy.

The American Agricultural Chemical Co. Grinding Plant — Pierce, Fla.

Kennedy-Van Saun 11' Radial Flow Classifier and Cyclone collectors.

IT COSTS LESS TO OWN THE BEST!

KENNEDY —

MANUFACTURING & ENGINEERING CORPORATION

Florida Phosphate Field

of The American Agricultural Chemical Co.!

OUTSTANDING FEATURES

- ***Lower Maintenance***

  Tramp iron, manganese, and other foreign material cannot damage system. *No magnetic separation required.*

  Grinding balls added while mill is in operation, maintaining constant level of grinding media. Production and fineness remain constant, month after month. No periodic shut-down for lubrication or replacement of wearing parts. Years of operation assured before parts (other than grinding balls) require replacement.

- ***Minimum Power***

- ***Lower Operating Cost***

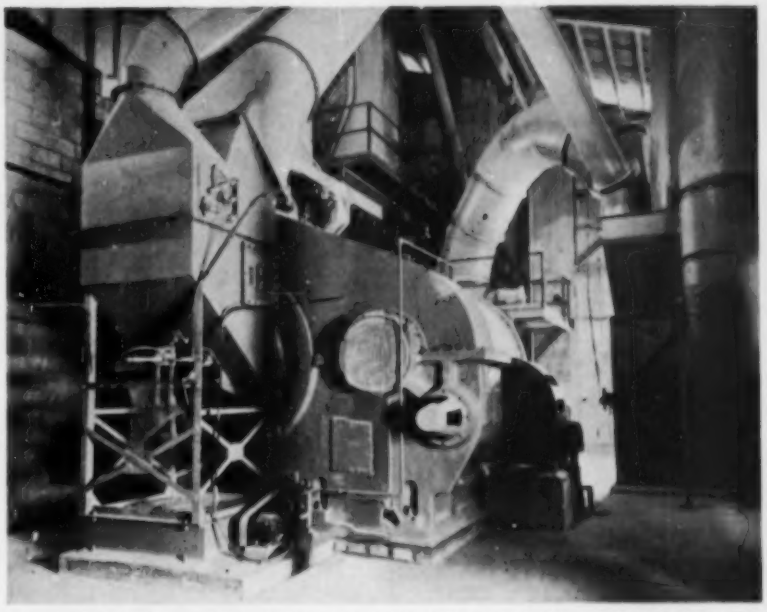
  Dependability of equipment and reliable automatic feed control assure high production with minimum operating personnel.

- ***Higher Production***

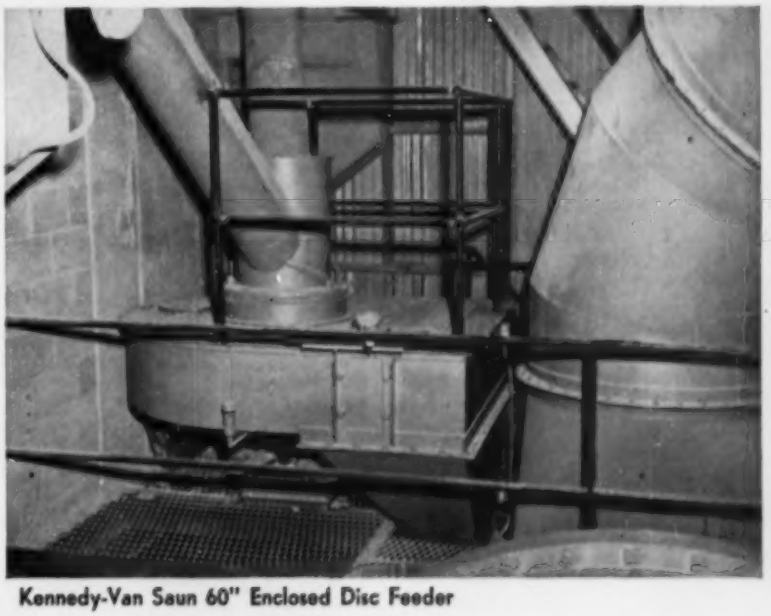
  Single grinding unit capacities to 75 tons per hour, requiring less floor space and structural work per ton of production.

- ***Flexibility of Design***

  Kennedy-Van Saun Air Swept Ball Mill Grinding Systems available in a wide range of capacities to meet your requirements.



Kennedy-Van Saun 10' x 15' Integral Gear Drive Air Swept Ball Tube Mill and #93 Exhauster Fan.



Kennedy-Van Saun 60" Enclosed Disc Feeder

VAN SAUN

Send for Bulletins describing KVS Machinery & Equipment

TWO PARK AVENUE, NEW YORK 16, N. Y. • FACTORY, DANVILLE, PA.

Personnel Service

(Continued from page 1050)

and surface survey work; graduate mining engineer preferred. Location, Canada. W4051.

Mining Engineers. a) *Mining Engineer-Geologist*, young, for exploration and development of mining properties. Latin American experience desirable. Salary, $4800 a year minimum plus extras. b) *Assistant Mine Superintendent*, with 10 to 15 years underground metal mining experience with major portion in Latin America. Salary, $6000 to $7200 a year plus extras. Location, South America. F4043.

Executive Assistant, 24 to 32, to take charge of office during absence of manager. Will dictate business correspondence, do sales work, and take occasional trips to Washington, D. C. Prefer graduate mechanical or mining engineer with some sales experience covering drills, hoists, and general construction and mining equipment. Salary open. Headquarters, New York. W4007.

Engineers. a) *Mining Engineer*, 35 to 40, with master's degree in business administration or the potential to absorb business aspects of mining operations. Should have some underground experience. Position is second to that of vice president and general manager of mining and smelting operation. Salary, $15,000 to $20,000 a year to start, plus company benefits. b) *Chief Exploratory Geologist*, to supervise staff of 9 to 12 geologists, for explorations in Guatemala, Mexico, and U. S. Salary open. Location, Midwest. W3988.

Junior Engineers, mining graduates, for sales application, production engineering or design, and development work with manufacturer of industrial, construction, and mining equipment. Salaries, $5000 to $6500 a year. Locations, various, in eastern U. S. W3965.

Mine Superintendent for Pb-Zn mine. Salary, $8100 to $8700 a year. Location, Mexico. F3800(b).

Geologist, graduate or equivalent, 35 to 45, with 10 years experience in mining geology; mine operating and field exploration work helpful. Will start at shaft operation, advance to field examination and charge of exploration department. Salary, $7500 to $8400 a year. Location, Rocky Mountains. S2259.

Assayer-Engineer, qualified to do fire and chemical analyses for common ores and make field examination of mineral prospects. Prefer single status; considerable travel. Salary, to $7000 a year. Position available April 1957. Location, Northwest. W4094S.

Mining Engineer, graduate, with at least five years good experience in underground mining and planning. At present operation is open pit but man must be qualified to go underground. Salary, $7200 to $8400 a year. Location, Midwest. W4055.

Geologists. a) *Exploration Geologist*, with experience, to supervise exploration project covering alluvial deposits. Salary, $8,000 to $10,000 a year. b) *Junior Geologist*, for field duties. Salary, $6000 to $8000 a year. Location, Liberia. F-4118.

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33 West 39th Street
New York 18, N. Y.

Please send me information pamphlet on services available, how air mail can expedite them, and their cost.

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Over Half Century Experience in

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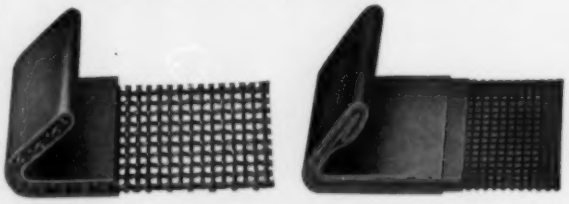
Reno, Nevada — Telephone FAirview 9-0732
Denver, Colorado — Telephone WEst 4-0673
Alaska — Contact Salt Lake City Office

Tyler Screen Sections for All Makes of Screening Machines!



Screen sections of Tyler Woven Wire are fabricated for all makes of vibrating screens in any mesh or metal. They are made up with hook-strip or bent-edge construction to suit the machine on which they are to be used.

Tyler rugged, accurately-applied hook-strips make possible stretching and maintaining the screens at drum-head tension, which is essential for successful screening and long screen life.

THE W. S. TYLER COMPANY
CLEVELAND 14, OHIO
Manufacturers of Woven Wire Screens and Screening Machinery
Canadian Plant—St. Catharines, Ontario, Canada

"Easy as filling a coffee bag," say men who prepare supply of Akremite.

Simple and safe to store, Akremite can be mixed right at the mine location.

"How We Cut Blasting Costs 50%, Increased Output 20%, with New Akremite Process"

Spencer prilled Ammonium Nitrate is base for new simple, safe, low-cost explosive

"We've found the new Akremite Blasting Process, now in use at our Cheshire, Ohio, operations, to be the ideal explosive for strip mining," reports F. H. Howe, Superintendent of Ohio River Collieries. "Because the ingredients are cheaper and the mixing process so much simpler, we have cut explosive costs 50% with Akremite. At the same time, better fragmentation produced by Akremite lets us move 20% more yards per month.

"The terrain in which we work is unusually difficult, but because Akremite is so safe and easy to handle, we still come out ahead."

☆ ☆ ☆

Akremite, the simple but revolutionary new explosive, is named after its inventor, R. I. Akre, superintendent of drilling at Maumee Collieries Co., Terre Haute, Ind. Akre uses prilled Ammonium Nitrate, produced by Spencer Chemical Company, as the base for his new explosive.

By itself, ammonium nitrate is not an explosive. But Akre found that if he mixed it with carbon black, packed it in a polyethylene bag, confined it in a drill hole and detonated it with a high gelatin dynamite and primacord, it produced a tremendous explosion.

Recently, an independent research organization put Akremite through a series of safety tests. They fired bullets into bags of freshly made Akremite. And they conducted the regular pendulum-friction and modified propagation-through-air tests.

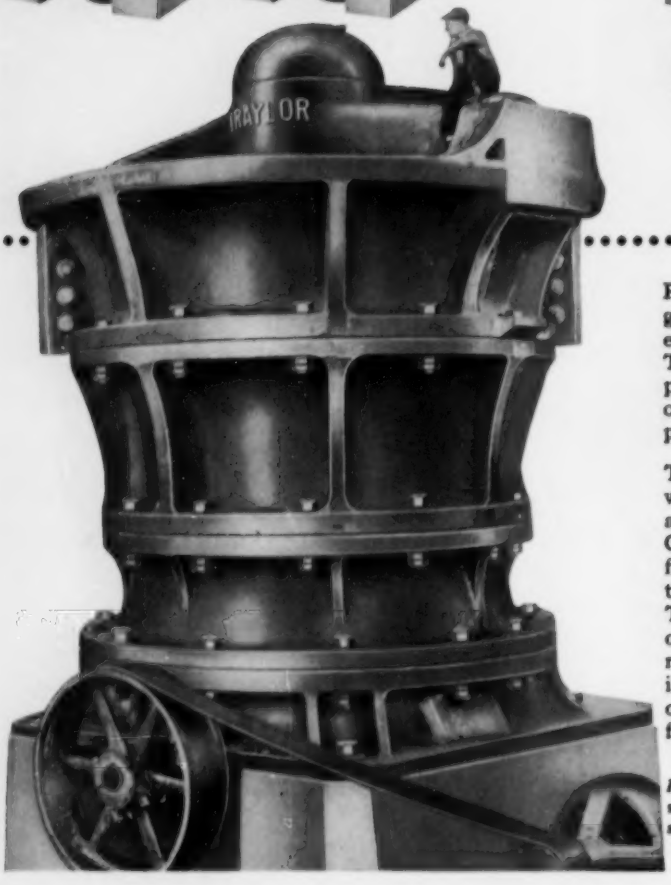
Not a single explosion, fire or crackling resulted from any of these tests. Akremite with Spencer prilled Ammonium Nitrate is not only cheaper and better, but many mines report it is the safest strip mine explosive they have ever used.

(NOTE: Spencer Chemical Co. will be happy to provide you with further information about the Akremite Method as discussed by Mr. Howe.)

SPENCER CHEMICAL COMPANY

610 Dwight Building • Kansas City 5, Mo. • BAltimore 1-6600

Number 6.. on the way!



Another Traylor TC Gyratory is added to the 5 on order for Big Taconite Project

Profitable production of iron ore from low-grade Taconite calls for the most modern, efficient methods and equipment. That's why Traylor Gyratories were the choice for both primary *and* secondary reduction at one of the extremely hard Taconite-bearing ore properties.

The initial order included a Gyratory Crusher with 60" receiving opening and 102" diameter crushing head, this huge TC Primary Crusher takes chunks of ore the size of a flat-top desk and reduces them to 12" at the rate of 4,000 long tons an hour. Four Traylor 36" Gyratories were included on the original order . . . to take the 12" ore and reduce it to minus 5" in the secondary crushing operation. Now . . . we've received an order for another 36" Gyratory to join the five now being built.

For complete specifications and description of the outstanding features of Traylor TC Gyratory Crushers, send for your copy of Traylor Bulletin #126.

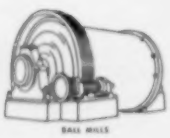
TRAYLOR ENGINEERING & MFG. CO.
857 MILL ST., ALLENTOWN, PA.
SALES OFFICES: New York · Chicago · San Francisco
CANADIAN MFRS: Canadian Vickers, Ltd., Montreal, P.Q.

Traylor

PRIMARY GYRATORY CRUSHERS

ROTARY KILNS

SECONDARY GYRATORY CRUSHERS


BALL MILLS

JAW CRUSHERS

APRON FEEDERS

Tumbling terrain buried more than 500 feet of 5-year-old Anaconda 3/0 Awg SH-D Shovel Cable. Despite accident, cable continued to supply power without interruption to giant shovel.

...and the wall came tumbling down

When this 90-foot wall came down unexpectedly, a five-year-old length of Anaconda 3/0 SH-D Shovel Cable was buried under thousands of tons of jagged rock.

Yet – *after ten days* – the cable and shovel it supplied continued to operate! Later, the cable was relocated–and is still giving service.

NEW BUTYL INSULATION

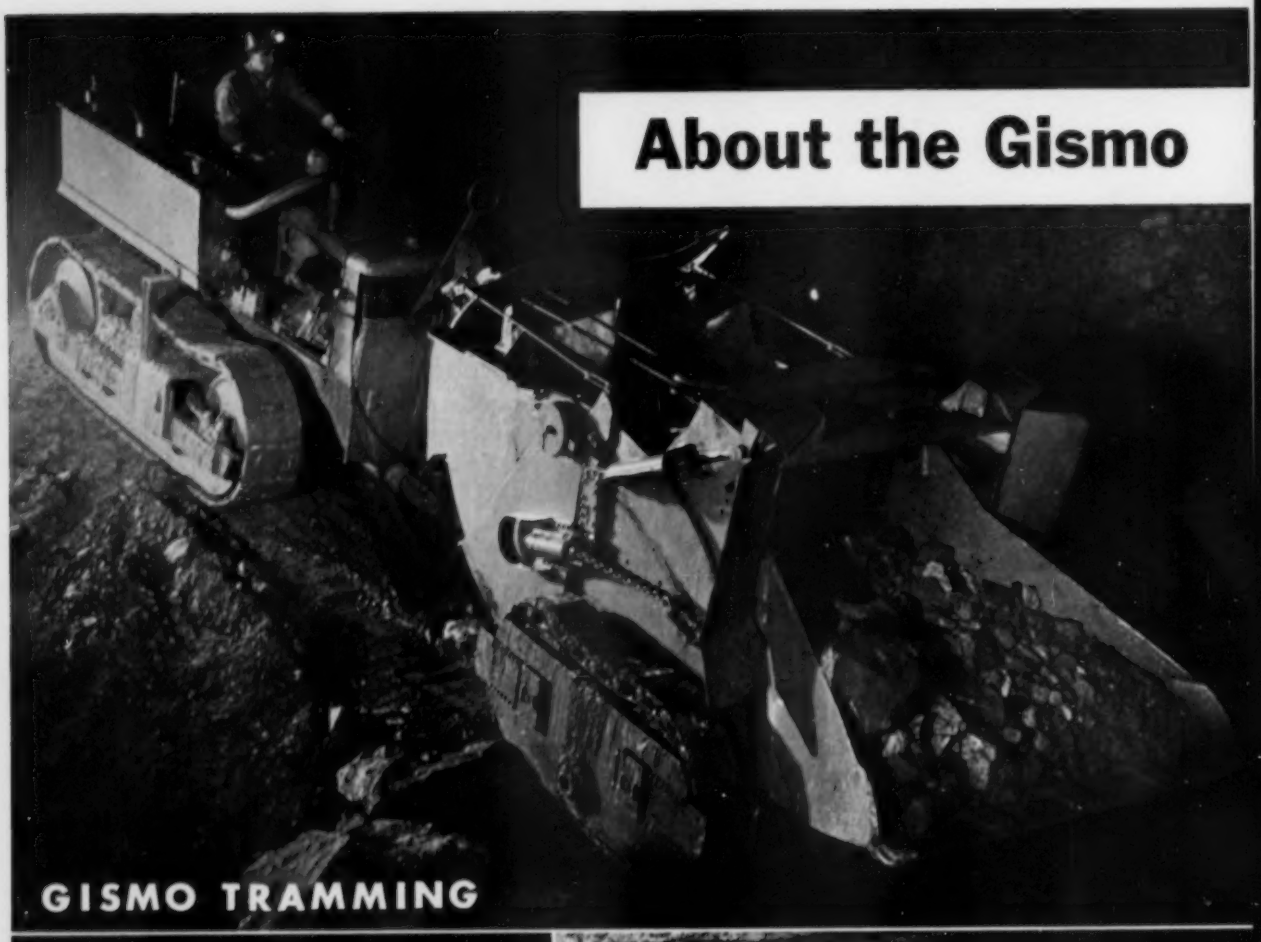
Longer life, better performing cable is the result of advanced engineering and manufacturing know-how that go into all Anaconda mine cables. Securityflex* Type SH-D Shovel Cable, for instance, is made with high-grade butyl insulation that withstands ozone, heat and moisture. Neoprene jacket is extremely tough and abrasion-resistant. Patented rubber cores cushion the ground wires and help prevent breaks.

Safety and economy of Anaconda cables have been proven in mines all over the world. The *Man from Anaconda* will be glad to help you on any cable problem. For cable, see your Anaconda distributor. And for copy of booklet, "*. . . and the wall came tumbling down,*" write: Anaconda Wire & Cable Company, 25 Broadway, New York 4, New York.

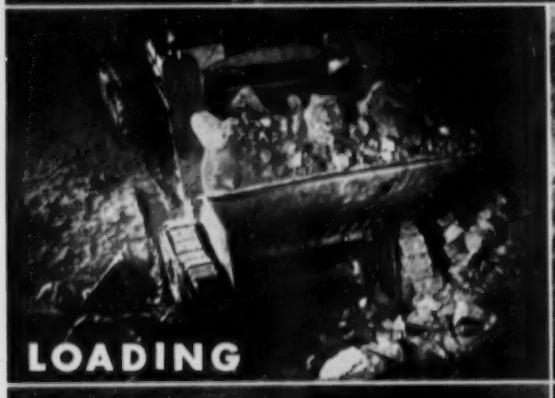
*Reg. U. S. Pat. Off. 8888-A

ask your distributor for ANACONDA® MINE CABLE

About the Gismo

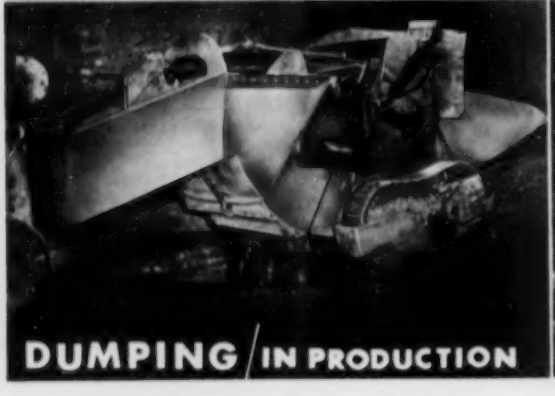


GISMO TRAMMING



LOADING

DRILLING



DUMPING / IN PRODUCTION

DUMPING / IN S-D CARS IN DEVELOPMENT

equipment– what are the facts?

Unquestionably, tomorrow's production economies and records in profitable mining operations will be the direct result of management's selection of proper equipment . . . equipment around which a simple production method can be devised. And certainly the major prerequisite to maximum simplicity is *minimum equipment, minimum labor with maximum production of both.* A plan conceived and organized by management around the criterion simplicity becomes increasingly efficient.

We believe the use of the Gismo equipment offers management the logical opportunity of devising a simple production plan. Quite successful systems organized and built around the Gismo have not only resulted in great economy in use of labor and equipment, but have practically eliminated deficiencies of time cycling operation.

Three men with one Gismo Self-Loading Transport, one Gismo equipped as a 4-drill stoping jumbo and a Gismo tractor have produced over a long period of time in excess of 400 tons per shift. Much less stoping space was required by the Gismo method. The simplicity of the operation together with the greatly reduced manpower made it possible for supervision to practically eliminate the problem of close co-ordination and time cycling between the various departments of the mine such as drilling, blasting, loading, transporting, hoisting and general services.

We realize producing such tonnage for so few dollars is a tremendous step forward. The natural reaction is to believe you are reading fantastic figures and statements. Nevertheless, all these facts are very true and they could mean but one thing. With the use of the Gismo equipment you are now free to organize the simple production method you have always strived for but have not been allowed to obtain because equipment permitting simple systems to function has not been available until now. *Sanford-Day Iron Works, Inc., Knoxville, Tennessee, USA*

We believe it is vital for mining management to become informed on how the Gismo equipment would fit into their operation. The advantages of this tremendous step forward have resulted in some companies abandoning well set plans to purchase other types of equipment in favor of investing in the Gismo. They found upon investigation, as we believe you, too, will find, that the economies offered by the Gismo equipment and the simple method of mining it enables you to organize are far greater—entirely too great to temporarily pass off simply as a new type of equipment. In fact, so great are its potential economies that this equipment could easily mean in but a few years the difference between profit or loss.

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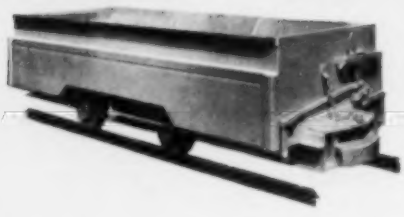

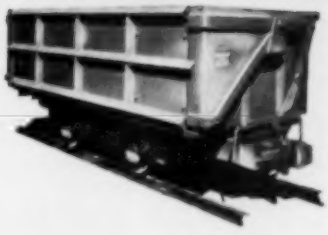

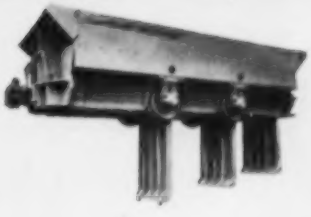

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BOOKS

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Clays and Clay Minerals, edited by W. O. Milligan, *National Academy of Sciences — National Research Council*, Washington, D. C., Pub. No. 395, 573 pp., $7.00, 1955.—The 43 papers in this volume are the proceedings of the Third National Conference on Clays and Clay Minerals held in October, 1954. Their subject matter includes geology, mineralogy, soil chemistry, soil physics, soil mechanics, physics, colloid chemistry, and ceramics. Illustrations, tables and charts support the several texts.

The Potentials About a Point Electrode and Apparent Resistivity Curves for a Two-, Three- and Four-Layer Earth, by Harold M. Mooney and W. W. Wetzel, *University of Minnesota Press*, Minneapolis, $18.00, 1956.—This work, of special interest to geophysicists, is presented in two parts. A 146 page bound volume sets forth all the tabular material with an introduction to its use and the mathematical basis for both tables and curves. The second section is an envelope containing apparent resistivity curves for about 2300 two-, three-, and four-layer horizontal structures. If purchased separately, the curves are priced at $15.00, and the text at $4.50.

Please Order These Publications from the Publishers

Iron Ore Deposits of Nevada, Bulletin 53, Part A, Nevada Bureau of Mines, *University of Nevada*, Reno, Nev., by Robert G. Reeves and Victor E. Kral, $1.00, 32 pp., 1955.—This bulletin, entitled *Geology and Iron Ore Deposits of the Buena Vista Hills, Churchill and Pershing Counties, Nevada,* is the first of a series still in preparation which will cover the full study of iron ore deposits in the state. The text is supported by eight maps, three tables, and 13 photographs, and was prepared in cooperation with USGS.

Instrumentation and Methods for Radioactivity Detection in the Mineral Industry, by James O. Milmoe and Stephen P. Kanizay, Vol. 51, No. 1, Quarterly, *Colorado School of Mines*, Golden, Colo., $1.00, 98 pp., 1956.—This Quarterly comprises two parts, the first of which deals with the theory, development, and applications of radioactivity detection instruments. The second outlines current exploration techniques.

The Geology of the Bennington Area, Vermont, by John A. MacFadyen, Jr., *Vermont Development Commission*, Montpelier, Vt., 68 pp., free, 1956.—Bulletin No. 7 of a series devoted to the geological study of various areas in Vermont, this report deals with the structure of the lower Paleozoic rocks in the southwestern section. The text is supported by maps and illustrations.

The Geology of the Lyndonville Area, Vermont, by John G. Dennis, *Vermont Development Commission*, Montpelier, Vt., 93 pp., free, 1956.—This study of the structural relations of the granitic plutons in the Lyndonville section is Bulletin No. 8 in the Vermont Geological Survey series. Illustrations and maps accompany the text.

The Development of Markets for New Materials, by E. Raymond Corey, *Division of Research, Harvard Business School*, Cambridge, Mass., 257 pp., $4.00, 1956.—This volume presents the results of a study of how markets are built for new end-products. The study is based upon a number of actual case histories of marketing procedures in the aluminum, fibrous glass, and plastics industries. It is designed to describe details of market development and to abstract and analyze some of the common problems encountered.

Geology and Mineral Resources of Nez Perce County, by Charles R. Hubbard, *Idaho Bureau of Mines and Geology*, University of Idaho, Moscow, Idaho, 16 pp., $1.00, 1956.—This is the first in a series of county reports to be issued by the Idaho Bureau of Mines and Geology on the geologic situation in each county of the state. It presents detailed information on the overall geologic environment and specific data about mineral occurrences of present or potential economic importance.

Geological Reconnaissance of the Cassia Mountain Region, Twin Falls and Cassia Counties, Idaho, by Walter Youngquist and Jerald R. Haegele, Pamphlet No. 110, *Idaho Bureau of Mines and Geology*, University of Idaho, Moscow, Idaho, 17 pp., 75¢, 1956.—This pamphlet presents the results of a brief survey of the Idaho section of the Cassia Mountain range, with special consideration given to Paleozoic stratigraphy and the nature and structure of Cenozoic sediments and volcanics.

Canadian Mines Handbook 1956, compiled, printed, and published by *Northern Miner Press Ltd.*, 116 Richmond St. W., Toronto 1, Canada, $3.00, 288 pp., 1956.—An up-to-date edition of the well known reference, part I lists principal companies including inactive and extinct companies; Part II, metal mines classified and mining stock prices.

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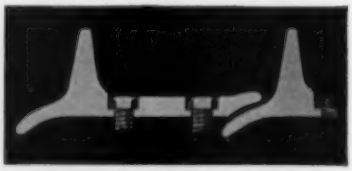

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Amsco Track Shoes have holes countersunk for less wear on bolts. Saves cost of replacing bolts when changing shoes.

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News
Equipment
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Sump Pump

Le Roi Div. of *Westinghouse Air Brake Co.* has a new air-operated sump pump rated at 340 gpm against a 10-ft head. Available in steel or bronze, the pump features low air consumption, a governor-controlled motor, and a built-in lubrication system. Priming is unnecessary and the 23-in. high pump will operate with a maximum head of 95 ft. **Circle No. 1.**

Dry Chemical Extinguisher

A new 30-lb automatic dry chemical fire-extinguishing system by *Ansul Chemical Co.* will provide protection at installations of ovens, stationary engines, small transformer

vaults. Automatic release (upper right) is activated when heat of a fire causes parting of a metal link located over the hazard. A CO_2 cartridge is punctured and pressurizes the extinguisher forcing its dry chemical charge into the piping (lower right) leading to nozzles directed at the trouble spot. **Circle No. 2.**

Wire Rope Clips

A 16-size selection of wire rope clips has been added to the *Canton Mfg. Co.* line of binding products. They are offered in three finishes: plain, cadmium plated, and hot dip galvanized. U-bolt portion is heavy steel with malleable saddle shaped to give maximum grip. **Circle No. 3.**

Small Compressor

The line of Gyro-Flo portable compressors by *Ingersoll Rand Co.* has an addition in a new smallest size, an 85-cfm unit weighing 1840 lb fully equipped on 2-wheel running gear. As a truck-mounted compressor, the GF85 weighs only 1375 lb and stands 42 in. high. A 4-cyl gasoline engine of L-head design powers both models and is equipped with a 6-v battery system and push button starting. **Circle No. 4.**

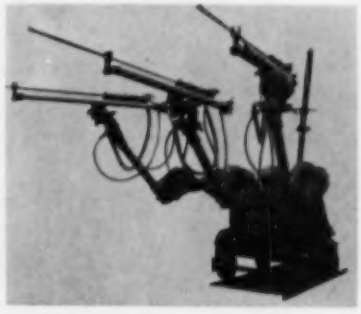
Bits for Bantamweights

Timken Roller Bearing Co. has new tapered socket carbide insert type bits with features claimed to give economy in the use of air-leg drills and light stopers. Deeper

wider clearance between bit wings and five-hole design make for faster chip removal, improve action, increase life. New carbides resist wear. Sizes are 1¼, 1⅜, and 1½-in. to fit ⅞ hexagon or ⅞ quarter octagon tapered drill steels. **Circle No. 5.**

Tri-Boom Drill Jumbo

One man can operate the three booms of a mine drill jumbo by *Thor Power Tool Co.* without leaving his station at the unit's air controls. The jumbo weighs 7800 lb fully equipped and requires only air hose hook-up to prepare for operation. Eliminated are time-consuming operations of wheel blocking, use of extension rods, and constant adjustment of the jack. The three booms can support power feed, chain feed, or air bar feed mountings. **Circle No. 6.**



Digger Teeth

Allied Steel & Tractor Products Inc. reports the use of new design principles in their Bulldog replaceable digger teeth for back hoe, dragline, clamshell, and loader buckets, and shovel dippers. New teeth are

attached by 1) burning off the old tooth to the adapter size, 2) welding on the adapter, and 3) securing the replaceable tip to the adapter by tapping pins in place with a hammer. Adapters are made in 15 widths from 2¼ to 7 in. and are claimed to save up to 65 pct on tooth maintenance costs. **Circle No. 7.**

Moto-Crane

A 35-ton Moto-Crane, model MC530W, has been added to *Thew Shovel Co.'s* Lorain line of power shovels and cranes. Powered by gasoline or diesel engines, the unit has standard power-assist steering.

Shear-Ball mounting of turntable to carrier eliminates adjustment and maintenance of center pins and nuts, and top or bottom rollers. Moto-Crane is fully convertible to crane, clam, dragline, or hoe. **Circle No. 8.**

News & Notes

Bradford Hills Quarry Inc. reports savings in operation and maintenance costs of chute liners by using T-1 plate steel manufactured by *Lukens Steel Co.* Used in a limestone chute where manganese steel plates stood up about eight months, the T-1 plates are still in service after over 17 months.

New Thor all-purpose one-man drilling machine

THOR introduces the powerful new 390 drilling machine with a host of features which speed up drilling operations and reduce operator fatigue.

This Thor drill is equipped with heavy-duty aluminum cylinder and chrome-plated steel piston rod and a specially designed 45 lb. sinker rock drill with integral feed leg connection. It can be easily converted for sinking. Thor 390 is equipped with a line oiler and automatic water valve which can be replaced with a plug for dry operation or collaring.

All controls are grouped on the backhead for safe convenient operation. A six position throttle, a Thor exclusive, gives a complete range of control from complete shut-off of air and water, to full air and water supply in logical steps for easy convenience and instant control.

Optional feed travel legs available in 36", 48" and the new model 680 72" telescopic leg.

Your nearest Thor representative will be glad to arrange for a demonstration. Thor Power Tool Co., Aurora, Ill.

MODEL 390
DRILLING MACHINE
MOUNTED ON 72"
680 TELESCOPIC LEG

Thor
PORTABLE POWER
TOOLS

New Thor 390—drifter, sinker, stoper—all in one

The Thor No. 390 is easily adaptable for all types of drilling at all angles. The drill can be removed from the leg for use as a sinker by loosening only one nut. Tool swivels into vertical position for stoping.

THOR POWER TOOL COMPANY

Atlanta, Birmingham, Boston, Buffalo, Chicago, Cincinnati, Cleveland, Denver, Detroit, Houston, Los Angeles, Milwaukee, Newark, Long Island City, N.Y., Philadelphia, Pittsburgh, St. Louis, San Francisco, Seattle, Toronto, Canada, Export Division, New York City

(21) GENERATOR SETS: Detroit Diesel Engine Div. of *General Motors Corp.* has a new brochure outlining its complete line of electric generating sets. Illustrated specifications are given of over 25 radiator and heat exchanger-cooled models ranging from 20 to 245 kw. Automatic starting and shut-down equipment is fully covered.

(22) ROOF BOLTING: A special issue of "Haulage Ways" published by *Ohio Brass Co.* offers 20 pages of the latest information on roof bolting techniques, including bolting materials, installation procedures, safety checks, pull tests. Included is complete information on O-B expansion shells and plugs and accessory bolting equipment.

(23) ROTARY FILTER: Features, design, operation, sizes, and capacities of a continuous vacuum-type horizontal rotary filter by *Dorr-Oliver Inc.* are detailed in a new 4-page bulletin. Intended for continuous dewatering and countercurrent washing, the filter consists of a circular horizontal filter table supported from beneath by a hardened ball race and rotated by a fixed chain drive.

(24) CONVEYOR LUBRICATORS: New 100 Series conveyor lubricators by *J. N. Fauver Co.* consist of three types for: trolleys and chains, bearings on slat-type conveyors, hot zone applications. Space is saved by compact aluminum housing, and crown filters and regulators have been improved.

(25) AUTOMATION ANSWERS: Bulletin PA561, an enlarged edition of "Proved Answers to Successful Automation," is offered by Photoswitch Div., *Electronics Corp. of America.* The 24-page brochure has detailed specifications and operational charts on packaged photoelectric systems for industrial control uses, including conveyor control and high-temperature measurement and control.

Free Literature

(26) FIELD TRIP FOOD: Prepackaged meals for field parties and expeditions are offered by *Bolton Farm Packing Co.* Not dehydrated, the food is heavy on calories and vitamins, and menus are varied. Packages are waterproofed and have a long shelf life.

(27) ROLL CRUSHERS: A bulletin from *McLanahan & Stone Corp.* gives data on Rockmaster single roll crushers. No backing and filling for a reversed stroke is one feature, and the crushers are said able to rapidly handle large tonnages of wet sticky feed.

(28) A-C RESEARCH: Facilities of the *Allis-Chalmers* Central Research Laboratories are shown and described in a 24-page digest. The creative role is stressed in "Research at Allis-Chalmers" and facilities covered involve electrical, mechanical, physical, nuclear, chemical, metallurgical, and processing research projects.

(29) PIPE PUSHING: Installation of pipes and conduits by forcing them underground endwise is described in a booklet by *Mercury Hydraulics Inc.* A gasoline or electric unit powers a hydraulic cylinder in the firm's Speed-Thru system which inserts pipe by a series of penetrating thrusts. Damage to roads, railroad tracks, and other surface installations is prevented because full-length trenching is unnecessary.

(30) BANTAM SPEED REDUCERS: In applications where a small but powerful speed reducer is needed, a choice of one of the Series 10 and 11 Bantam units by *Metron Instrument Co.* may suit the job. Three classes provide over 600 ratios.

(31) X-WELD CHAIN: Accoloy X-Weld 125 chain is a new product of American Chain Div. of *American Chain & Cable Co. Inc.* said to resist bending, breaking, and kinking. Weld area of 2¼ times that obtained with regular welding methods is claimed to produce more than double security at the weld. Bulletin DH-319 gives details with a section on specially manufactured heat-resisting and acid-resisting chains and studs that also have the X-Weld feature.

(32) COAL FLOCCULANT: *The Dow Chemical Co.* has a technical manual, "Separan 2610 in the Coal Industry," that describes specific uses of their flocculating agent. Applied to coal processing, Separan is reported to have increased settling rates 30 to 40-fold and boosted filtration rates as much as 40 to 60 pct.

(33) METAL BUILDINGS: A color catalog from *Stran-Steel Corp.* details a newly enlarged line of industrial and commercial buildings and accessories. Use is made of Stran-Satin metal wall for good appearance combined with strength in a modern rib design. Buildings are available in widths of 32, 40, 50, 60, 70, and 80 ft, or multiples of these widths, in bays 20 ft long.

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for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

11

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51	52	53	54	55	56	57	58	59	60
61	62	63	64	Students should write direct to manufacturer.					

(34) TITANIUM TUBING: Seamless titanium metal pipe and tubing for pressure and mechanical uses is offered by Tubular Products Div. of *The Babcock & Wilcox Co.* Technical Data Card 185 gives condensed data on sizes and tolerances, mechanical and physical properties, corrosion resistance, fabrication, and heat treatment.

(35) DIESEL ENGINES: Nordberg 2-cycle heavy-duty diesel and Duafuel engines are the subject of new 20-page Bulletin 245 from *Nordberg Mfg. Co.* The TS-21 engines are the port scavenging and port exhausting type and are available in a range of 5 to 12 cylinders of 21½ in. bore. Variable pilot oil timing in Duafuel engines is designed to provide gas ignition timed for best combustion at a specific load. Duafuel engines are convertible to oil or natural gas operation.

(36) SPRAY VALVES: *The Farval Corp.* has an 8-page brochure with information on spray valves for open gearing and slide surfaces. Valves provide proper coating of gears, in measured amounts, at regular intervals. Complete enclosure of the system keeps lubricant clean at the nozzle orifice. Booklet offers schematic layouts and application views.

(37) VARI-SPEED DRIVES. Featured in a new bulletin from *Worthington Corp.* are motor pulleys (⅛ through ¾-hp) available with adjustable and tilting motor bases, companion sheaves, and V-belts. Pulleys have oil-impregnated bronze bushings—require no lubrication.

(38) TRACTOR ATTACHMENTS: *Hyster Co.* booklet 1305 outlines methods of increasing productivity through the use of Hyster job attachments on tractors, new and old. Accessories described include the Hystaway excavator-crane, grid roller, Tractor Donkey, yarders, and winches. Pointers are prepared from on-the-job case histories.

(39) HEAT EXCHANGERS: Alcotwin fin-tube heat exchangers are described in a new 8-page bulletin from *Alco Products Inc.* Detailed are the heat-transfer characteristics of Alco longitudinal fin tubes, which feature U-shaped fin channels integrally bonded to the tubes by heliarc welding. Included is a table of dimensions, a type designation table, and finside coefficient and finside pressure-drop charts for all sizes of exchangers.

(40) LAB SPATULA: *Fisher Scientific Co.'s* Chemiscraper is a large polished stainless steel spatula intended for laboratory use. Ground thin at both ends, the tool has flexible blades for efficient mixing and blending of materials. Two lengths are available—12 in. and 18 in.

(41) COMPRESSORS: On-the-job photos, diagrams, and complete specifications of the complete *Clark Bros. Co.* line of compressors, engines, and turbines are given in a 26-page bulletin. Included is a new combustion gas turbine, said to be the first unit of its kind designed specifically for pipeline use.

(42) D-C MOTORS: *Allis-Chalmers Mfg. Co.* has a new bulletin on d-c motors, ½ through 200 hp, and motor-generator sets of up to 200 kw output. Covered are shunt and stabilized shunt wound, compound wound and series wound motors in eight different enclosures plus a matching line of motor-generator sets for conversion of a-c power to d-c.

(43) DISC ROLL MILL: Bulletin 52 from *Hardinge Co. Inc.* details features of an improved disc roll mill intended for relatively soft materials that grinds by means of two rolls riding on a horizontal disc or table which rotates at low speed. Some advantages claimed are quiet operation, flexibility of feed size, wide range of product fineness, lubrication without shutdown, minimum floor space requirements, tramp iron freed without causing damage.

(44) CHART PROTECTION: Dropping a chart or blueprint into a transparent acetate envelope will keep it usable longer. Instructions stay readable and work flows smoothly without interruption or the need for costly reproducing. *American Kleer-Vu Plastics Inc.* offers a sample holder and a catalog of acetate envelopes for industrial use.

(45) STOP & GO TRANSMISSION: For automotive equipment in stop and go service, the *Clark Eqpt. Co.* TransVerter provides torque converter, hydraulic clutch, and transmission in a compact package unit. Hydraulic disconnect clutch allows provision for one of a variety of clutch releases including shift lever button. Many load variations can be met without a gear shift and the torque converter gives smooth starts —no stalling of engine or slipping of clutch.

(46) SLURRY PUMP: *Dorr-Oliver Inc.* has a diaphragm slurry pump designed to operate at pressures up to 100 lb and powered by compressed air, vacuum, or both. Available in six standard sizes with capacities ranging from 5 to 90 gpm, the ODS pump is claimed to handle slurries containing crystals or floc without causing damage to the product. It has been applied to a wide variety of abrasive slurries and solutions—both hot and cold, acid and alkaline.

(47) SAVING THROUGH STANDARDS: A 40-page survey report from the *American Standards Assn.* highlights economic benefits to industry resulting from the use of standards. The report, "Dollar Savings Through Standards," covers 79 case studies of 27 industrial fields, including mining and conveying machinery. Information for the booklet was obtained by ASA through a special survey among its trade association, technical society, and company members.

(48) SPEED REDUCERS: *Westinghouse Electric Corp.* offers a new booklet, the "Speed Reducer Engineering Manual." Manual presents complete application information on hp ratings, torque capacity, overhung load values, installation and maintenance for single, double, and triple reduction. Parallel shaft horizontal speed reducers are available in ratings from 1 to 4000 hp in six standard designs. Within certain limits, they may be used for speed increasing service.

(49) PIPE WRAPPING: Techniques for protecting underground pipe from corrosion with "Scotchrap" brand pipe insulation tape is the subject of a new booklet from *Minnesota Mining & Mfg. Co.* The 24-page manual provides step-by-step photos and instructions for wrapping bends, elbows and short sections, patching pipe with tape, preparing joint surfaces, wrapping welded joints and other typical fittings.

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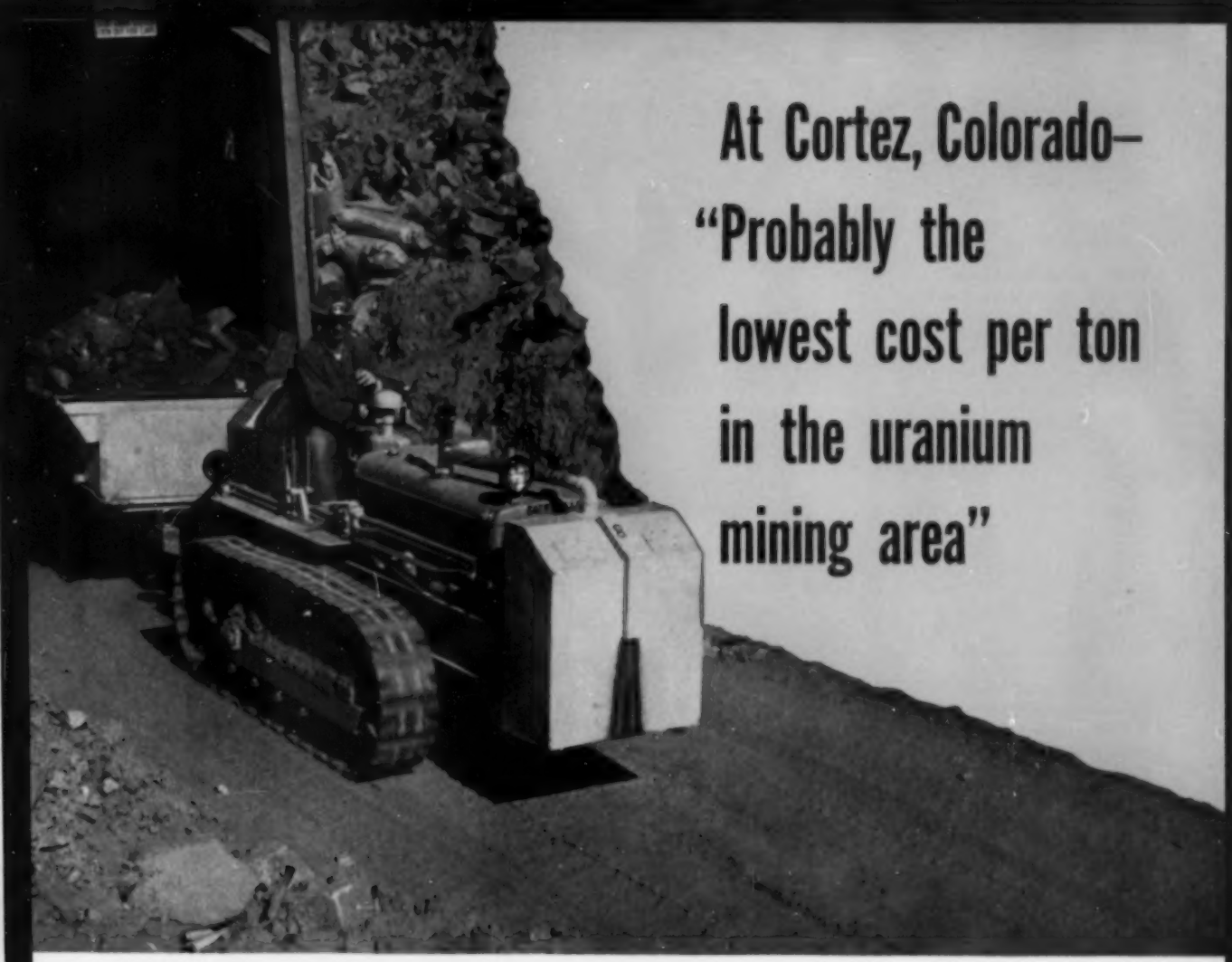

At Cortez, Colorado– "Probably the lowest cost per ton in the uranium mining area"

Archie Garwood is a partner in G. & G. Mining Co., of Cortez, Colo. Here is what he says about the firm's eight Caterpillar track-type Tractors: "We chose Cat-built machines because we knew we'd be getting the best. Our operating costs are very low—probably the lowest per ton in the entire uranium mining area. Although we're operating in one of the most inaccessible localities in the U.S., we've had excellent service from our Caterpillar Dealer."

Two CAT* D2 Tractors with tram wagons average about 360 tons per day hauled out of the mine. Another D2 is available for the same work if needed, and a drill jumbo is mounted on a fourth. Notice the exhaust scrubber mounted on the front of the D2 Tractor shown above. Thanks to clean combustion afforded by their Caterpillar-built fuel injection systems, Cat diesel-powered machines with scrubbers are entirely practical for underground use.

Three Caterpillar-built Traxcavators* load the ore underground, and above ground where trucks haul it to Moab, Utah, 45 miles away. A Caterpillar D6 Tractor is used for 'dozing haul roads, stockpiling, and general clean-up work around the mine.

The compact and maneuverable D2 Tractor is ideal for underground use, and wherever working space is cramped. Now available with optional oil clutch for longer facing life and fewer clutch adjustments, the new Caterpillar D2 Tractor is built to outwork and outlast any other machine in its class. With a wide choice of attachments to choose from, the versatile D2 can be "tailor made" for use in *your* mine.

Your Caterpillar Dealer has full details on this tough and willing worker. Ask him for an on-the-job demonstration—and count on him for prompt service and parts you can trust.

Caterpillar Tractor Co., Peoria, Illinois, U.S.A.

CATERPILLAR*

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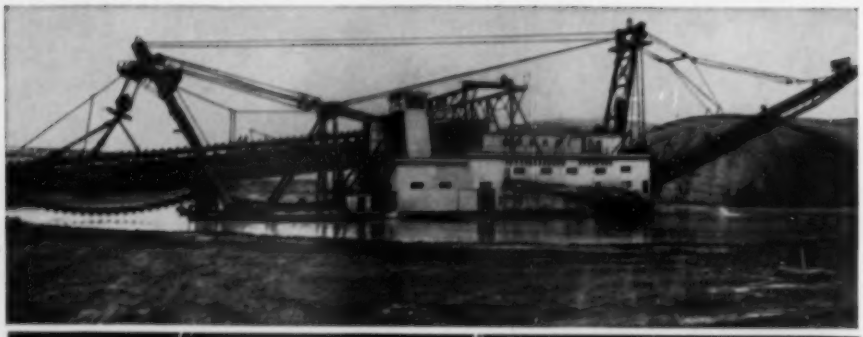
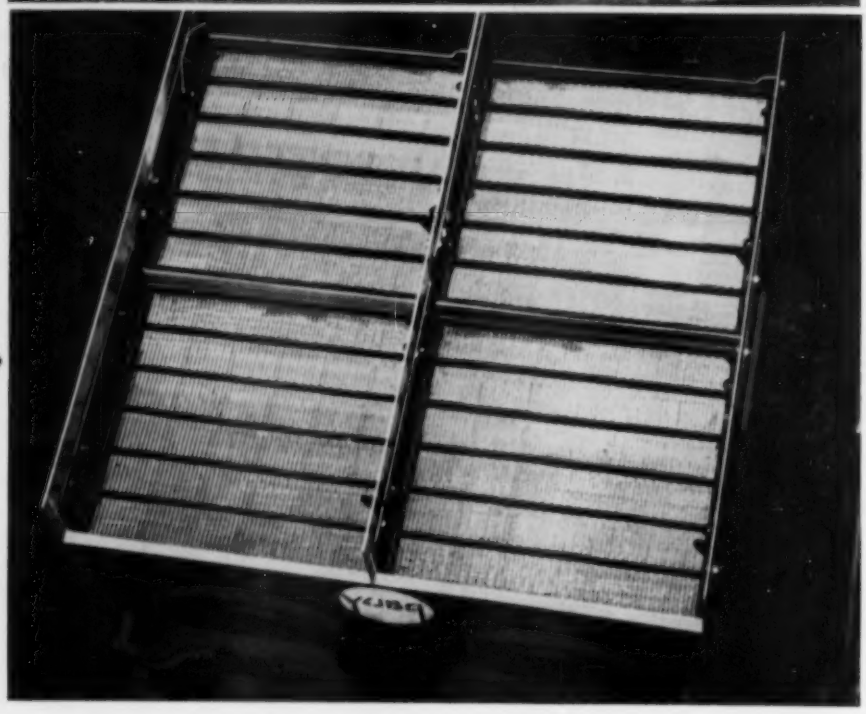
NAME THE DATE... YOUR DEALER WILL DEMONSTRATE



YUBA Gold Dredge floats on self-dug, land-locked ponds. 4 of these dredges can move about 13,000 cubic yards of placer material a day. Pay dirt is separated from waste through stainless jig screens.



Type 316 Stainless Steel Jig Screens solve abrasion and corrosion problem in Yuba gold dredging operation. Even after 7 years, screen stays open, resists rust, wear, clog-up. Mineral Jig built by Yuba Manufacturing Co., Hammonton, California.

Stainless steel screens going great shakes for YUBA

Last seven years and more. No rust. No clog-up.

Water . . . black sand . . . gold-bearing gravel . . . they're all in the day's shake of mineral jigs like this.

Yet Yuba Manufacturing Co. hasn't replaced its stainless steel screens in the last seven years. What's more, they're *still* in good condition.

"Just never wears out", says H. A. Sawin, Asst. Director of Sales, speaking of the austenitic chromium-nickel steel wire used.

Think of what that means in replacement costs . . . in downtime saved when Yuba operates as many as 48 of these 4-screen assemblies on a single gold mining dredge!

Could be you'll make a strike with stainless in your ore processing equipment. It's light. It's strong. It effectively resists corrosion and abrasion. It pans out in performance.

You can choose from a wide variety of stainless steels to meet tough corrosion and abrasion problems. We will be happy to make a few suggestions on the right alloy for you. Just write.

INCO TRADE MARK **THE INTERNATIONAL NICKEL COMPANY, INC.** 67 Wall Street New York 5, N. Y.

Asarco to Search Northern Rhodesian Area

American Smelting & Refining Co. has taken an option on the 1000 sq mile Lunga concession in Northern Rhodesia from New Discovery Mining Corp. New Discovery will do the exploratory work for Asarco, with particular attention to a limonite deposit expected to require deep core drilling.

National's New Mining Job on Schedule

Construction at National Potash Co.'s property near Carlsbad, N. M. is reported on schedule with shipments due in February 1957. Shafts are down to ore level at the $19 million project which is jointly owned by Freeport Sulphur and Pittsburgh Consolidation Coal Corp.

Kaiser Confirms Interest in Florida Phosphate Byproducts

A pilot plant on the property of Virginia-Carolina Chemical Corp. will be installed by Kaiser Aluminum & Chemical Corp. to examine economic possibilities of extractions from phosphate wastes. Speculation is that search will center on recovery of fluorine containing substances.

Sudbury District Gets Iron Ore Project

Moose Mountain iron ore mine 35 miles north of Sudbury, Ont. is to be reopened as an open pit operation with concentrator by Lowphos Ore Ltd. M. A. Hanna Co. is agent for Lowphos which is wholly owned by National Steel Corp. Goal is half million tons of concentrates per year by 1958. Final 1955 figures show Canadian iron ore production reached 14.536 million tons, 121 pct over 1954, and valued at $110 million. Most of tonnage was exported.

GSA Programs Purchases for Three Ores

Up to 250,000 tons of newly mined acid grade fluorspar and up to 2,000 tons of top grade chrysotile asbestos will be purchased by GSA. Termination date for both programs is Dec. 31, 1958 and both apply to ores of domestic origin. . . . Also programmed is purchase of up to 250,000 lb of contained combined pentoxide in columbium-tantalum bearing ores and concentrates of domestic origin.

Aluminum Enters Structural Applications

Harvey Machine Co. has formed a new division to engineer, fabricate, and erect aluminum structures. Company says that "for first time aluminum structurals comparable in size and cost to steel (e.g. wide flange beams over 16 in. depth) have been made available for immediate delivery." The structures division with headquarters near Los Angeles will use the production and fabrication facilities of Harvey Aluminum.

Illustration from Agricola's De Re Metallica (1621)

Arduous Work—Small Production

In centuries past, the production of metals could be measured by handfuls. Small wonder when comminution was done by a hammer and beneficiation was limited to hand-picking.

Fortunately, today's modern processing plants treat many tons of ore and produce many tons of metals, thus making possible our "metal age" civilization. This large production requires modern grinding methods which in turn utilize such grinding media as those produced by CF&I. The grinding balls and grinding rods produced by CF&I are *always* made from special analysis steels with the ideal balance between toughness and hardness to assure optimum grinding ability and maximum wearing ability.

THE COLORADO FUEL AND IRON CORPORATION CF&I 3695

Albuquerque • Amarillo • Atlanta • Billings • Boise • Boston • Buffalo • Butte • Casper • Chicago • Denver • Detroit • El Paso • Ft. Worth
Grand Junction • Houston • Lincoln (Neb.) • Los Angeles • New Orleans • New York • Oakland • Oklahoma City • Philadelphia • Phoenix
Portland • Pueblo • Salt Lake City • San Antonio • San Francisco • Seattle • Spokane • Wichita • CF&I OFFICE IN CANADA: Toronto

Powerful crowding action fills this big, half-yard capacity bucket in any muck pile. Fast discharge and near-perfect clean-up at the face are time-saving abilities of the Eimco 630.

An Eimco 630 "moves in" for another load, operated by easy-to-reach, easy-to-work controls.

The Eimco 630 Crawler-Excavator is gaining more respect for its adeptness in a greater variety of underground mucking tasks by responding favorably to new tests of its versatility.

One recent field report points to an Eimco 630 Transport-Loader that has been loading rounds much faster and getting better "clean-outs" during five months of competition with a 48" box scraper with a 50 h.p. slusher.

Near-perfect clean-out by the 630 increases drilling efficiency. To get 630 clean-out results in a slusher operation, requires use of snatch blocks along the foot and hanging walls and final clean-up at the face thru hand shoveling.

So outstanding has been the Eimco 630's performance, future stopes are being planned for maximum utilization of its many operating advantages.

The 630 Transport-Loader with integral, automatic discharge, 30 cu. ft. hopper, is very profitably in use for short hauls and floor-level dumping in a wide variety of industries.

And—like all 630 excavator-dozer units, the transport loader has that extra maneuverability thru independent track operation—control simplicity and the big, half-yard bucket capacity.

BEFORE YOU DECIDE ON A MINING PATTERN, investigate the EIMCO 630. You'll find—as others have—it pays to plan your mining strategy around operating advantages of the versatile 630.

THE EIMCO CORPORATION

Salt Lake City, Utah—U.S.A. • Export Offices: Eimco Bldg., 52 South St., New York City

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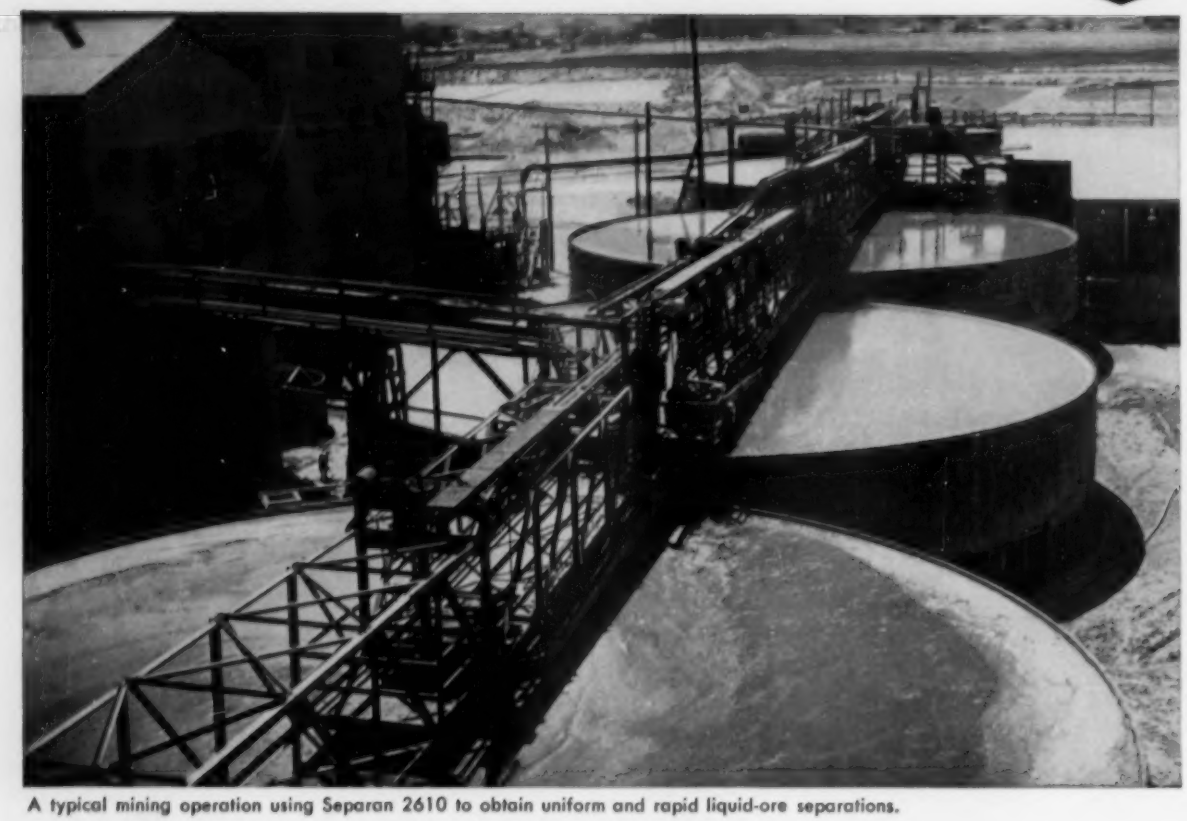
B-4219



A typical mining operation using Separan 2610 to obtain uniform and rapid liquid-ore separations.

FLOCCULATING POWER, EASE OF USE, ECONOMY, CREATE BIG DEMAND FOR SEPARAN 2610

Outstanding new efficiency can be brought to settling and filtration operations. That's the reason for the soaring demand for Separan* 2610. Exciting case histories are piling up.

One Thickener for Five

One uranium ore processor was able to eliminate four of his five thickeners with the addition of Separan 2610. Two banks of filters and several earthen ponds were also no longer needed. The savings? Nearly $3000 per day. In another uranium mill, production was at 50% of capacity due to low filtration rates. But when 0.2 pound of Separan 2610 per ton was added, the filtration rate doubled—and so did production!

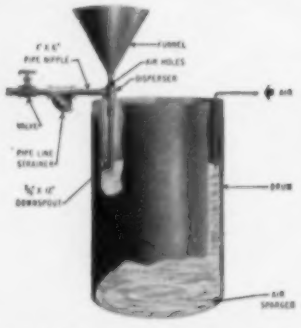
Settling Faster, Filtration Better

The advantages of using Separan 2610 are numerous. Overhead clarity is increased, materials lost in overflow decreased. In filtration, this synthetic non-ionic polymer increases cake weight per unit area of filter and reduces moisture content. Washability is increased, the cake is more homogenous and easier to handle.

Dow Xanthates—Dowfroth 250

Separan 2610 is the latest addition to the Dow family of chemicals for mining. Dow Xanthates, the collectors to use in all flotation of sulfide minerals, and Dowfroth® 250, the frother for improved metallurgy with reduced frother consumption, are two other well-known Dow mining chemicals. For a sample or information on any of these products write to THE DOW CHEMICAL COMPANY, TECHNICAL SERVICE AND DEVELOPMENT, Midland, Mich., Dept. SC926D-1.

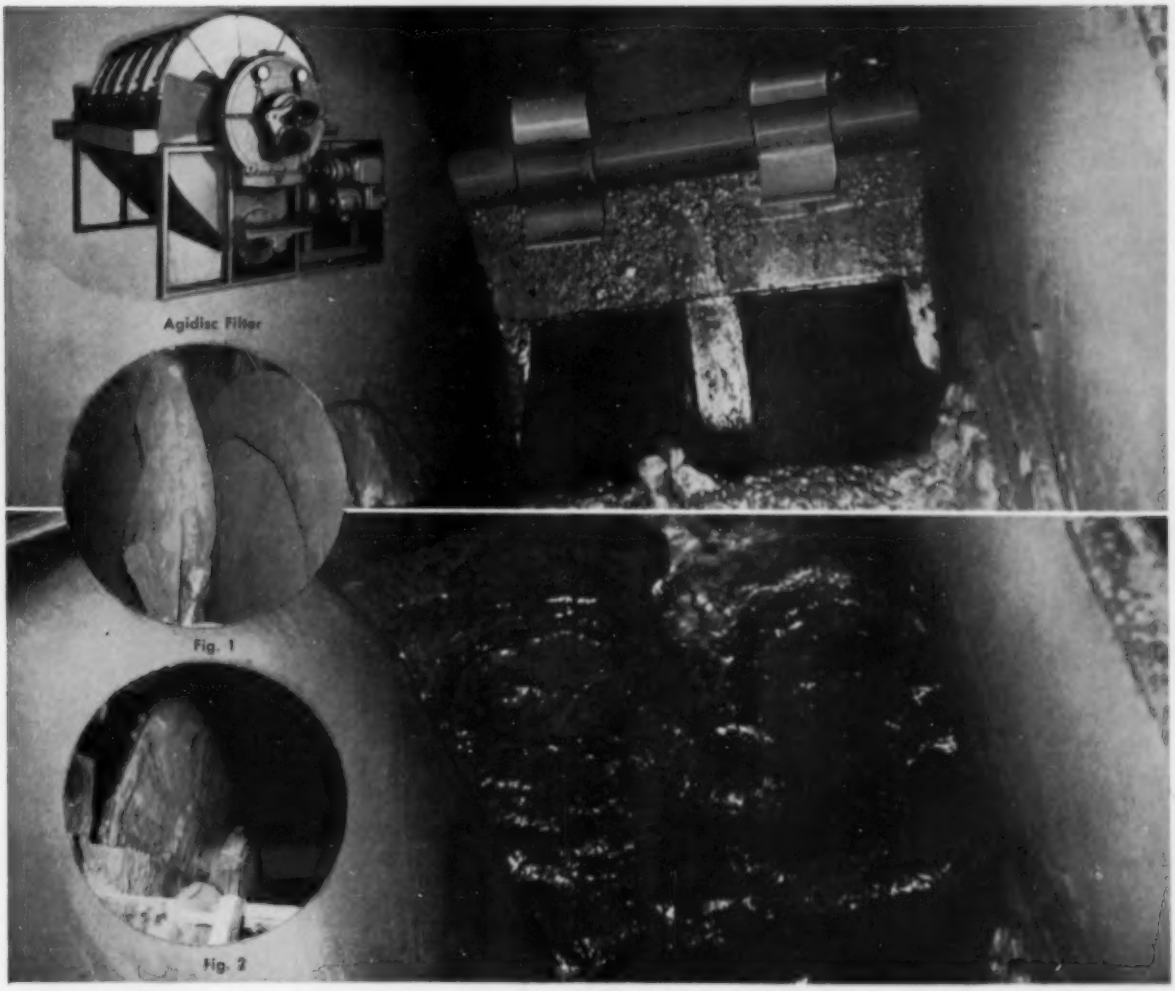
*Trademark



New Disperser has been developed by Dow to permit dissolving of large amounts of Separan 2610 without use of a mechanical mixer. The solution needs no preservative in normal storage or use.

you can depend on DOW CHEMICALS

DOW



THE EIMCO AGIDISC FILTER GETS UNIFORM RESULTS

The Eimco Agidisc is the only disc type filter with proven ability to get uniform cake formation from all fast settling solids.

A veteran superintendent of a concentrator, processing complex metallurgical slurry says, "The Eimco Agidisc is the one really effective means to get uniform particle suspension in disc filter tanks."

This magnetite pelletizing plant uses five 6 X 6 Eimco Agidiscs. Each filter is capable of processing 640 long tons of concentrate per 24 hour day. The feed contains 63% solids at 62% minus 325 mesh. Specific gravity is 5.0 The filter cake has 7.5% moisture.

Experience has taught filter station personnel that **uniformity** is the key to good operating results when fast settling particles are in the feed. Only the Eimco Agidisc Filter gives strong, properly directed agitation

Insufficient agitation results in a classification ring on the periphery of the disc with thin, slimy cakes near the center. Disadvantages here are high cake moisture and low capacity. (See Fig. 1).

If agitation is too violent, the result is a scouring action on the periphery of the disc. Disadvantage here is principally low capacity. (See Fig. 2).

The Eimco Agidisc eliminates both of these costly formations as shown in figure 3. (One disc is removed to show construction). A variable speed reducer promotes proper surface movement. The Eimco gets **uniform** cake dryness; **uniform** cake thickness; **uniform** particle dispersion and **uniform** tonnage consistent with feed.

The Eimco Agidisc can help your operation by increasing tonnage and lowering moisture content in the cake. Write for complete information today!

THE EIMCO CORPORATION

Salt Lake City, Utah—U.S.A. • Export Offices: Eimco Bldg., 52 South St., New York City
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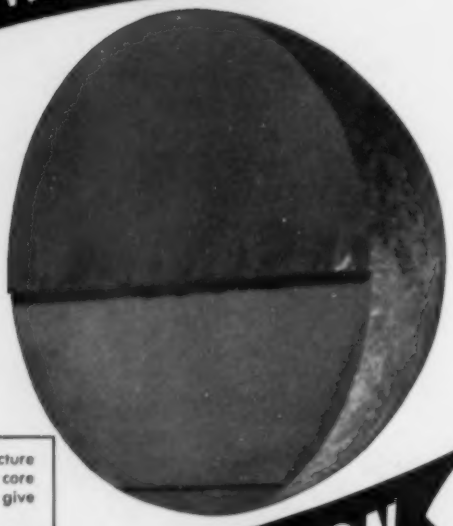
B-221

For Over 20 Years the World's



Fine, even-grained structure extends deep into the core of Moly-Cop Ball to give long, even wear.

STANDARD of COMPARISON

SHEFFIELD

ARMCO®

MOLY-COP

TRADE MARK

COPPER-MOLYBDENUM-ALLOY

Grinding Balls

SHEFFIELD STEEL

DIVISION

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EXPORT REPRESENTATIVES

THE ARMCO INTERNATIONAL CORPORATION, Middletown, Ohio

EIMCO 105 -- MORE POWER FOR MORE WORK

Eimco 105 Tractor-Excavators are balanced to give maximum work efficiency in every phase of digging and loading. Compare these figures—(A) 39,200 pounds of digging force at the bucket lip as the 105 moves into the rock pile; (B) 39,200 pounds of lifting capacity for breakout power.

This power gives the operator the same potential in productive capacity as he would have if you bought him a boom type shovel costing three to four times as much.

How does Eimco design a small (1½ yard) Tractor-Excavator to outproduce every other machine in its price range?

The answer to this is Eimco's unique Tractor design which provides better balance, lower center of gravity and delivers full engine horsepower to the bucket at all times.

Eimco also makes it easier to operate the 105 Tractor-Excavator. The operator sits up front where he can see what he is doing. Two small handles, easily held in one hand, control all movements of the Tractor. The operator does more work with less effort, stays efficient the entire shift.

Other firsts in the Eimco 105 Tractor-Excavator include (1) independent track control so that one track can be run forward while the other turns reverse; (2) separate final drives for each track; (3) full track oscillation on the tractor when equipped with loading or excavating attachment; (4) elimination of master clutch and drag-track steering; (5) Unidrive transmission in which gearing always rotates in the same direction; (6) all alloy steel construction; (7) clutches that never need adjustment — and many other exclusive features.

See these completely new tractors as Bulldozers, Excavators or Loaders working near you. Write for complete information.

THE EIMCO CORPORATION

Salt Lake City, Utah—U.S.A. • Export Offices: Eimco Bldg., 52 South St., New York City

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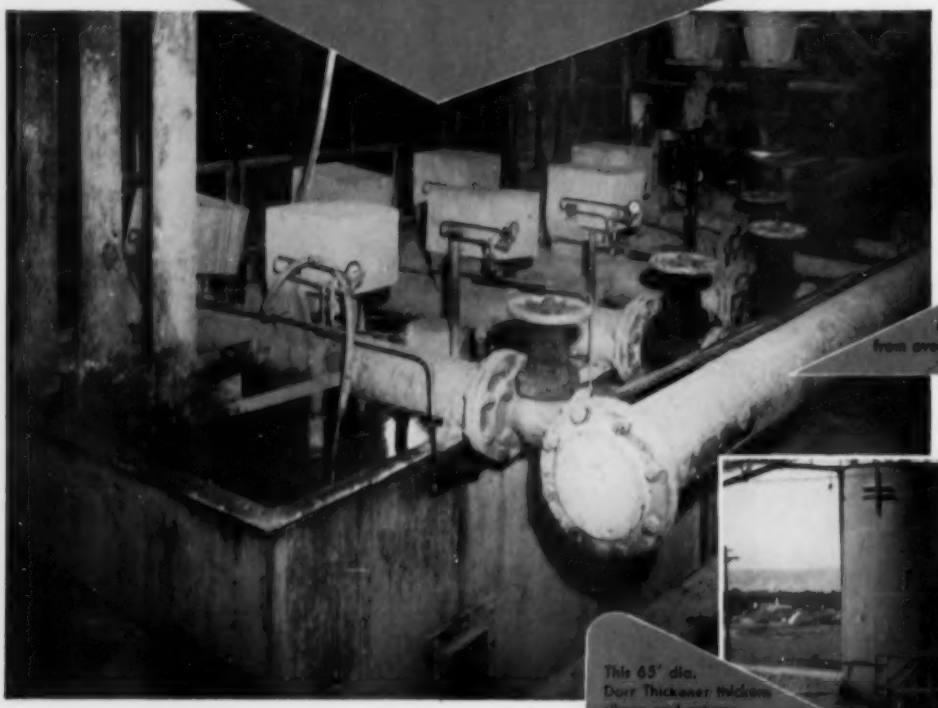
DORRCO *JET* SIZER

at Climax Uranium...



Closeup of Dorrco Jet Sizer at Climax Uranium taken from overflow end of machine.

This 65' dia. Dorr Thickener thickens slimes and returns overflow for re-use.

Earliest producer on the Colorado Plateau, Climax Uranium uses Dorr-Oliver equipment for key classification, thickening and filtration steps throughout the flowsheet. The initial sand-slime separation is made in a 7½ pocket Dorrco Jet Sizer. Feed to the unit is 400 TPD of all 28 mesh material which is split at 150 mesh. A hindered settling classifier, the Dorrco Jet Sizer utilizes an overhead, removable hydraulic water distribution system which eliminates costly constriction plate construction. Adjustments to changes in feed composition are automatic to insure constant pulp density and product quality.

Sizer spigot product goes to leach tanks and sizer overflow (−150 mesh) to a 65′ dia. Dorr Thickener. Thickener overflow is returned for re-use and underflow is sent to process. A 50′ dia. and 24′ dia. Dorr Thickener are also installed at Climax along with a number of Dorrco VM Pumps.

In the leaching section an 11′6″ x 12′ Oliver Panel Filter performs a liquid-solids separation on the leached uranium ore. Employing a floating cake discharger, the Panel Filter is ideal for handling slow filtering products forming slimy cakes.

For more information on the complete line of D-O equipment for the mining industry, write for a copy of bulletin No. 2200. Dorr-Oliver Incorporated, Stamford, Connecticut

DORR-OLIVER
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WORLD-WIDE RESEARCH • ENGINEERING • EQUIPMENT
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In Minnesota Taconite...and the World Over

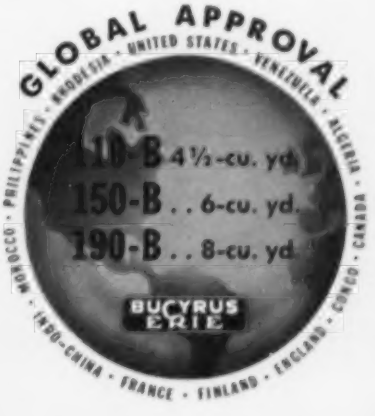
Bucyrus-Erie Ward Leonard Electric Excavators Help Boost Output the Low-Cost Way

Swinging an 8-yd. dipper, this Bucyrus-Erie 190-B shovel is loading taconite into ore cars near Virginia, Minn. Like other Bucyrus-Erie Ward Leonard electric shovels, it has advanced-design features that keep production up and costs down.

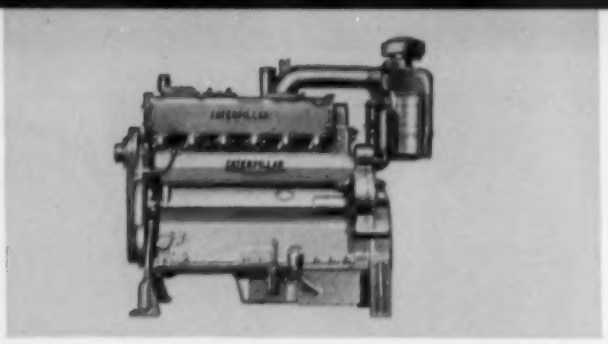
Its front-end design provides plenty of strength without deadweight, to move more material with less effort. Ward Leonard control assures rapid acceleration and deceleration for high-speed, coordinated operation. There is extra torque and ample usable power whenever needed.

In every feature, from boom point to treads, there's strength for low maintenance and maximum time on the job. Let us give you the facts on the right size Bucyrus-Erie shovel for keeping output up and costs down in your mining operations.

94L56C



BUCYRUS-ERIE COMPANY •
• South Milwaukee, Wisconsin



A QUARTER CENTURY OF

A CATERPILLAR FIRST
—the "Hi-Electro" hardened cylinder liner

A CATERPILLAR FIRST
—the chemically conditioned cylinder liner

A CATERPILLAR FIRST
—the stainless-steel piston protector

A CATERPILLAR FIRST
—the steel-backed aluminum bearing

A quarter of a century ago, Caterpillar created mobile diesel power. For the first time, the power of the diesel engine was unleashed from its bulky foundations and put to work in the field—compact, economical. Here was diesel power of simple design, with no need for experts to operate and maintain. Here was diesel power with the lugging ability to knuckle down to the tough jobs.

The introduction of mobile diesel power was a tremendous advance in many fields. It provided efficient diesel power for tractors, motor graders, earthmoving equipment . . . for the work boat, the gin, the locomotive, the oil rig, the municipal plant . . . for *any* application in which steady, low-cost power is crucial. And everywhere, CAT* Diesel Engines proved themselves durable and dependable. They established Caterpillar as the leader in diesel engineering.

Today, hundreds of thousands of modern heavy-duty Cat Diesels are on the job in every corner of the world. And still the research continues. Study and experiment go ahead constantly in Caterpillar laboratories. Special testing machines help point the way toward new advances. Manufacturing techniques improve, too, in the world's largest diesel engine factory—where the quality of workmanship is the standard for the industry.

A modern world must have modern power—more and more of it. It is coming, in ever increasing quantity, from the production lines of Caterpillar, the leader.

Caterpillar Tractor Co., Peoria, Illinois, U. S. A.

CATERPILLAR*

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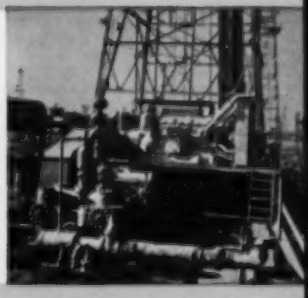
DIESEL POWER FOR PROGRESS



DIESEL LEADERSHIP

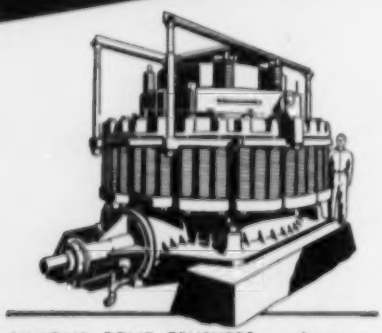
A CATERPILLAR FIRST
—interchangeable, adjustment-free fuel injection equipment

A CATERPILLAR FIRST
—the capsule-type injection valve

A CATERPILLAR FIRST
—the service meter

A CATERPILLAR FIRST
—superior lubricants (detergent oils)

This view, showing a specimen of an etched cross section of a molybdenum ingot, was furnished by Climax Molybdenum Co., who now use a 60" Symons Primary Crusher, 12 Symons Cone Crushers, and 25 Symons Vibrating Rod Deck Screens.



SYMONS CONE CRUSHERS . . . the machines that revolutionized crushing practice . . . are built in Standard, Short Head, and Intermediate types, with crushing heads from 22 inches to 7 feet in diameter — in capacities from 6 to 900 tons per hour.

• MOLYBDENUM, from the Greek "*molybdos*", meaning lead, is popularly referred to as "moly", and has its main commercial source in the mineral molybdenite. Increasing production of molybdenum has been essential for the manufacture of alloy steels used extensively in aircraft, automotive, heavy machinery, and many other industries.

Largest producer of "moly" in the world is Climax Molybdenum Company, supplying well over half of the free world production. Located high in the Rockies in Colorado, this miracle mine of modern times is presently the largest underground mining operation in North America. At Climax, a battery of twelve 7-ft. Symons Cone Crushers handle the reduction crushing of the vast tonnages of hard crystalline rock bearing the molybdenite ore.

For in molybdenum production . . . *as in all the great ore and industrial mineral operations the world over* . . . Symons Cone Crushers are the outstanding choice of leading producers for processing great quantities of finely crushed product at low cost.

Nordberg Mfg. Co., Milwaukee, Wisconsin

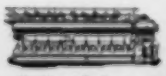
SYMONS . . . A REGISTERED NORDBERG TRADEMARK KNOWN THROUGHOUT THE WORLD.

Symons Gyratory Crushers

Grinding Mills

Mine Hoists

Symons Vibrating Grizzlies and Screens



Nordberg Engines 10 to over 12,000 H.P.

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MACHINERY FOR PROCESSING ORES and INDUSTRIAL MINERALS

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Crusher Razed At Cliffs Shaft

Skillful Blast Topples Old Column

Timber-r-r . . . bellowed the surface foreman with all the enthusiasm of a lumberjack as his well-planned blast toppled hundreds of tons of concrete pillar to earth. A final shot had been fired in the razing of the old shaft mine crusher of Cleveland-Cliffs Iron Co., Cleveland. Buildings and installations painfully close by stood undamaged as the 72-ft crusher stretched its length on the very line planned for its fall.

The 46-year-old pillar had posed a tricky demolition problem. Adjacent installations included a shaft-house, loading pocket ramp, machine shop, and railroad tracks. All were within the circle of potential damage if the 12x11x72-ft giant did not fall to the planning of the shaft mine's surface crew. A heavy wire cable was attached to the top of the pillar, and a bulldozer well out of blast-range exerted pull in the direction the structure was to fall. Twelve ft holes were drilled into the concrete base. Shoe by shot, the foreman and his crew of four men proceeded to undercut the pillar base in the direction of the tractor's pull. With the firing of shot number five, the massive column shook and started a slow lean earthward. Gaining momentum, it struck the earth and raised a heavy dust cloud. Its tremendous weight drove it a foot into the ground and caused it to split into huge blocks at each pour line.

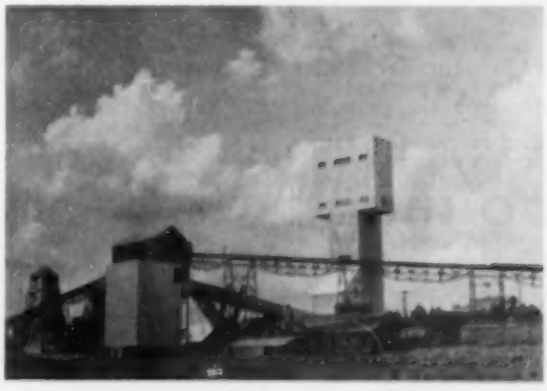
Debris was cleared by blasting the blocks and carrying the broken material off in crane-loaded heavy trucks.

Now, geared to a new shaft and Koepe hoist, a new underground system has replaced the old crusher. For many years it had been a familiar part of the skyline at Cliffs shaft mine.

Charges are set to gradually undercut the base of the old crusher.

Crusher gives way, in the direction of a tugging cable, to the blast of the final charge.

First unit crashes to earth and splits into blocks at pour lines.



The new Cliffs Shaft plant.

RESERVE PROJECT WORKS

MESABI TACONITE

Pelletized Concentrate Produced

Reserve Mining Co.'s interest in taconite began in 1939, the year the company was formed, when certain leases were acquired in the huge deposits located in the Mesabi Range of Minnesota. Today, Reserve holds a deposit that is some nine miles long, 2800 ft wide, and 175 ft deep at the widest point. Virtually free of glacial overburden, the area contains an estimated 1½ billion tons of magnetic taconite which can be mined by the open pit method. Since three tons of taconite are required to produce one ton of pelletized concentrate, a potential 500 million tons of high grade iron ore exist in the holdings. Geological studies also point up the possibility of substantial additional tonnage.

Taconite has proved to be a strong adversary for mining men. The problems lay partly in mining the tough mineral, but processing proved even more difficult.

In 1942, experimental work by Reserve personnel was begun in collaboration with E. W. Davis who was director of the Mines Experiment Station, University of Minnesota. Early in 1948, the first iron made from taconite alone was produced. Later, developments led to the construction, by Reserve, of a processing plant at Silver Bay and Babbitt, Minn. Designated the E. W. Davis Works, the plant produced its first pellets in the fall of 1955. On April 10, 1956 production at Silver Bay passed the million-ton mark.

Three-in. taconite arriving at Silver Bay goes to car dumper building (top), on to storage bins, and then to fine crusher building (left).

The E. W. Davis Works has an annual rated capacity of 3¾ million tons of pellets per year. The plant includes a giant coarse crusher at Babbitt, in which primary and secondary crushing is done. Also at Babbitt are loading bins and railroad repair shops for Reserve's 47-mile interdepartment railroad connecting the Babbitt portion of the works with the Silver Bay facilities.

At Silver Bay are located: facilities for dumping cars; fine-crusher storage bins; a building where third and fourth-stage crushing is done; and a 1307-ft long concentrator building which houses 12 units for grinding, magnetic separation, and concentrating. The 1400x185 ft pelletizing plant at Silver Bay contains six units which include balling drums and pelletizing furnaces. A large ore bridge supplies the pellet stockpile, and storage bins on a dock feed pellets to two loaders which convey the pellets into ore-carrying vessels.

SKINNER DRYERS
AT LARGE COPPER COMPANY

- each drying approximately 20 tons/hr. of copper sulphide concentrates . . .
- reducing moisture content from approximately 15% to 9%.

Two Colorado Iron Works 23' 6" I.D. 4 hearth Skinner Dryers

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Skinner . . . proved for 37 years for efficient roasting and drying of ores, clays, limestone, lime sludge . . . including sulphide roasting; oxidizing, reducing and chloridizing roasts.

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CYANAMID

REAGENT NEWS

"ore-dressing ideas you can use"

Gold Operators: —Flocculate Slimes *And* Prevent Deposition of Lime Salts With AEROFLOC® Reagents

AEROFLOC Reagents not only serve as powerful flocculants for slimes, they also are very effective in sequestering or complexing calcium, thereby preventing deposition of troublesome lime salts.

This double benefit from the use of the AEROFLOC Reagents has been proved at several cyanidation operations. AEROFLOC 548 and 552 appear to be most effective for this double use. Unlike the polyphosphates which sequester calcium but disperse pulps, the AEROFLOC Reagents sequester calcium and *flocculate* pulps.

A Cyanamid field engineer writes concerning a western United States cyanidation operation, using 0.03 lb. per ton of AEROFLOC 548 Reagent:

> *"Subject has had no trouble with lime deposition in bags and pipes, etc., since using AEROFLOC. In this CCD plant AEROFLOC 548 gives pulp density of 42% solids and clear thickener overflow, as against 25% solids and dirty overflow without AEROFLOC."*

Besides flocculating and settling slimes, AEROFLOC 548 and 552 Reagents may also perform the following useful functions at your plant:

1. Prevent blinding of filter cloth caused by lime salts precipitating in the fibers, resulting in lower labor costs, fewer acid washes and longer cloth life.
2. Prevent formation of incrustations of lime salts which cause stoppages in pipes, launders, jigs, deaerating equipment, etc.
3. Lower calcium content of gold precipitate with resulting increase in gold content, cutting down fluxing and melting costs.

For further information on this useful application of the AEROFLOC Reagents, contact any Cyanamid field engineer or Cyanamid Mineral Dressing office.

AMERICAN CYANAMID COMPANY

MINERAL DRESSING DEPARTMENT

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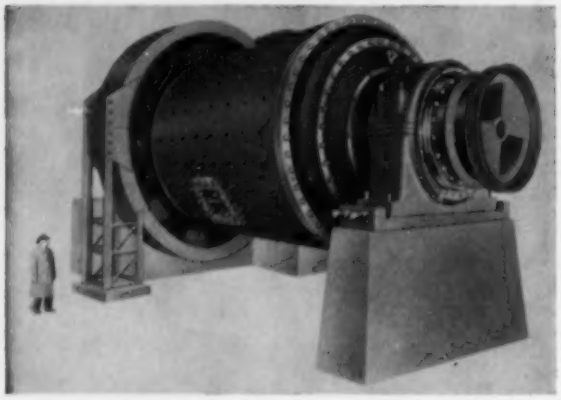
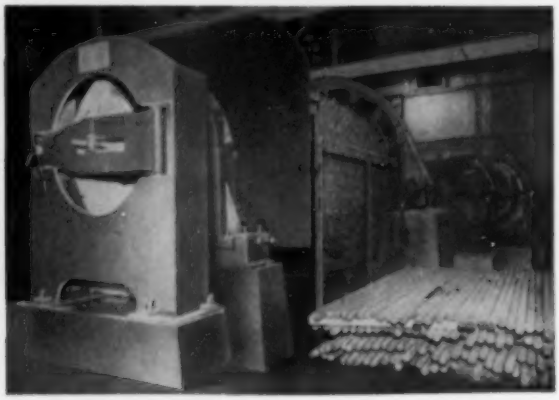
MINE & SMELTER COST CUTTING EQUIPMENT

MARCY GRATE DISCHARGE BALL MILLS AND MARCY OPEN END ROD MILLS

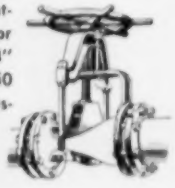




Marcy Ball and Rod Mills both incorporate the Marcy principle of grinding..."rapid change of mill content is necessary for high efficiency." This is accomplished by use of the Marcy *full-grate* discharge on ball mills and the *open end* feature on Marcy rod mills. These features result in a low pulp line which provides an active, effective grinding mass to act on particle size reduction only...there is no wasteful cushioning action by high pulp levels. There is a faster migration of fines than oversize particles, thus less overgrinding. This basic principle of grinding incorporated in Marcy Mills has proved, in hundreds of installations, to give greater output with lower KWH per ton, from 25 to 45% more tonnage compared with the same size overflow mills. Marcy Mills are made in sizes up to 12'6" diameter. Every Marcy is backed by more than 40 years grinding experience.

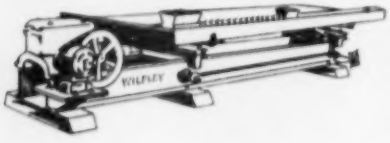
MASSCO-GRIGSBY RUBBER PINCH VALVES

Designed for abrasive and corrosive pulps. Patented hinged sleeve for longer wear. 1" to 14" diameter. Up to 150 pounds continuous pressure.



WILFLEY TABLES

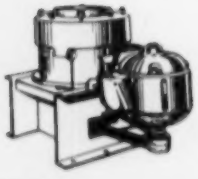
For separation of any ore or material amenable to gravity concentration. Laboratory and commercial sizes, up to 75 tons capacity per 24 hours.



MARCY PULP DENSITY SCALE

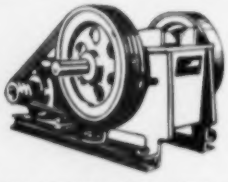
Gives DIRECT READING of weight of sample in grams or kilograms; specific gravity of liquid, specific gravity of pulp; percent solids contained in a pulp of any given specific gravity; specific gravity of dry solids. Is very accurate and easy to keep clean.

LABORATORY EQUIPMENT



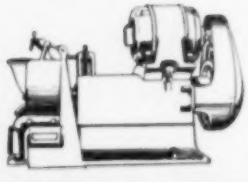
MASSCO GY-ROLL REDUCTION LABORATORY CRUSHER

Reduces ½" feed to as fine as 10 mesh in single pass. High capacity with low power consumption. 6" and 10" sizes.



MASSCO LABORATORY JAW CRUSHER

Welded steel frame; manganese steel jaw and check plates; bronze bushed bearings; smooth jaws give better product and easier cleaning. Adjust for plate wear by convenient hand wheel.



MASSCO-McCOOL PULVERIZERS

Disc type grinder with planetary movement. No gears. Will grind to 150 mesh in one pass.

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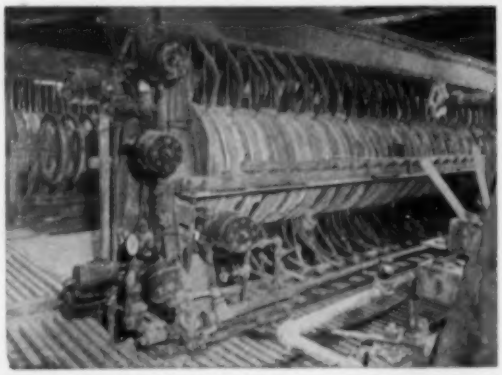
Holes Jet-Pierced

Difficulties in mining caused by the hardness of taconite have caused modifications of the open-pit method. After overburden has been stripped by bulldozers and power shovels, blast holes are sunk in the taconite by jet piercing machines. Conventional drills can sometimes do no better than one ft per hr in the tough material. Jet-pierced holes (of about 7-in. diam) are loaded with explosives and the taconite is blasted loose.

Heavy electric-powered shovels are used to load the taconite into side-dump trucks which haul the rock out of the mine to the coarse crusher. When crushed to —3 in. the ore is shipped by rail to Silver Bay. There further reduction is followed by magnetic separation. Ball mills are used for still finer grinding and the ore passes through another series of magnetic separators. Remaining material is then washed in a hydro-separator to remove silica and the result is a jet black mineral about as fine as cement. Agglomeration follows and the resulting pellets are fired at 2400°F to harden them for handling.

All operations at the plant are planned to run year-round, with no shut-down in freezing weather. During the off-season the pellets are stockpiled. When boats are loading at the dock during the shipping season, the pellets from the plant and from the stockpile will be loaded simultaneously by a system of belt conveyors.

Aerial view of Reserve Mining Co.'s taconite processing facilities at Silver Bay, Minn. In the background is the community of Silver Bay, expected to become next largest town on north shore of Lake Superior, after Duluth. Steam marks the pelletizing building. Pellet stockpile area and storage bins are seen in foreground.



DeZurik Plug Valves with air operators on an Eimco Burwell filter

DeZURIK
PLUG VALVES

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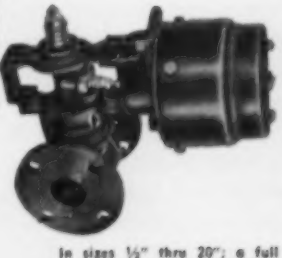
NON-LUBRICATED VALVES
FOR SO MANY
METALLURGICAL SERVICE LINES!

The *exclusive* eccentric principle of DeZurik Plug Valves guarantees friction-free operation, dead-tight shut-off and low maintenance without lubrication for a wide choice of service lines.

Basically DeZurik's eccentric principle consists of a resilient-faced metal-cored plug with an eccentric shape and an eccentrically-raised seat. Contact between the plug face and the seat is made ONLY when the valve is fully closed. In opening the valve, the slightest rotation of the plug breaks the contact and it swings wide open in a quarter-turn—smoothly, easily, without friction! Tight sealing and easy operation are obtained *without lubrication*! And despite scale, dirt, or other solids in the flow . . . DeZurik Valves shut tight!



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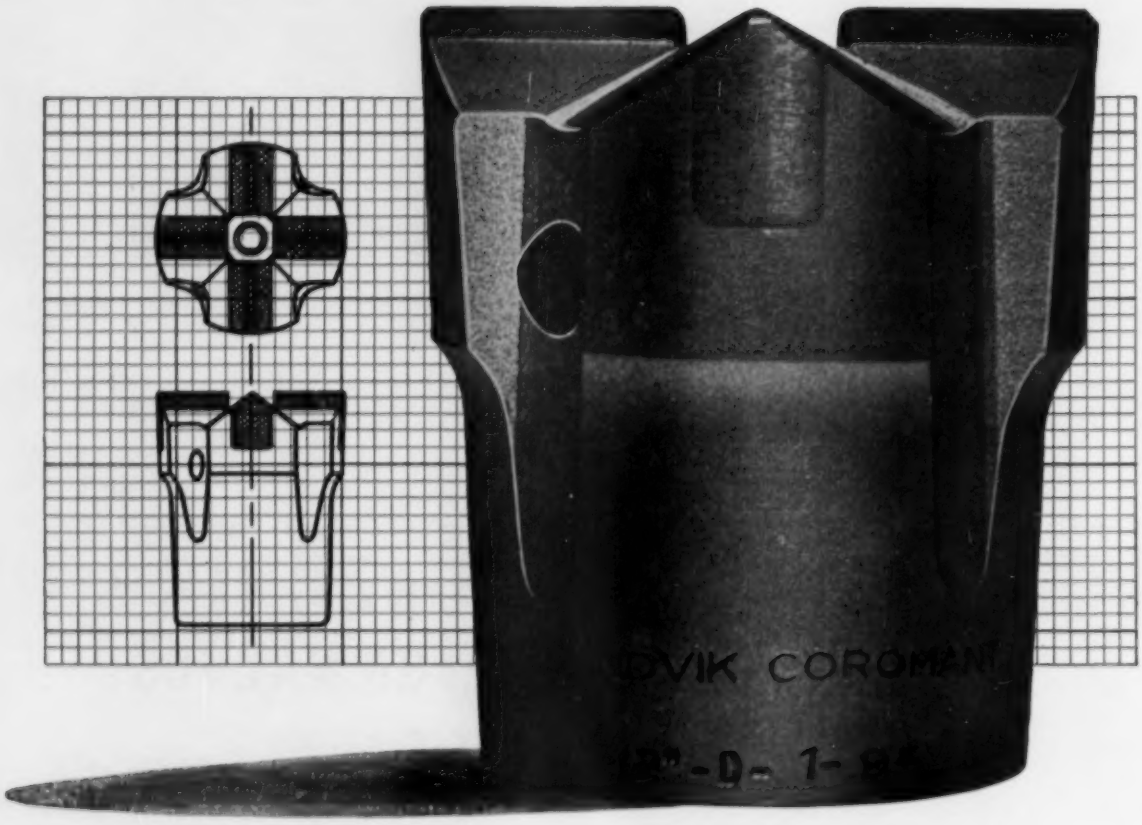
In sizes ½" thru 20"; a full range of metals and models.

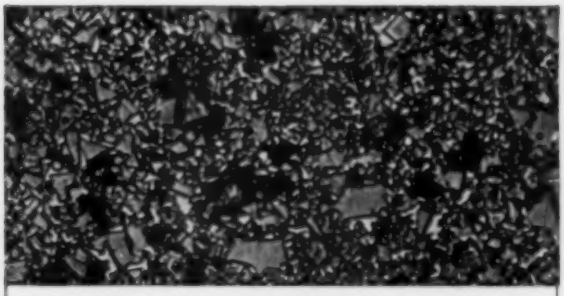
THIS ROCK BIT IS PRECISION-MADE FOR A HIGHER PERFORMANCE



Nothing tougher and more wear-resistant than the insert of a Sandvik Coromant 776 bit

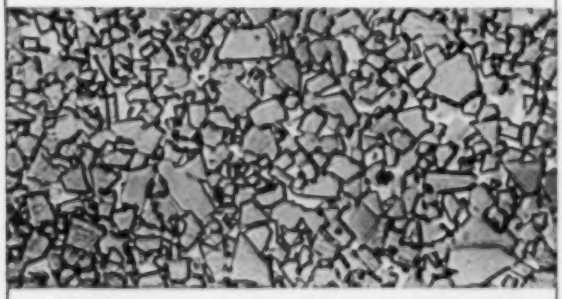
Rock bits that go on *and on* must have highest-grade tungsten-carbide inserts. Nothing but tungsten carbide in its purest state is good enough, will last as long. That's why the carbide that goes into a Sandvik Coromant 776 bit is meticulously controlled.

Sandvik, the world's largest manufacturers of brazed-in tungsten-carbide inserts for rock drilling, control every phase of production. Coromant carbide is scrutinised for impurities from the very first stages of processing the tungsten ore, right through to the final inserts. Add to that Sandvik's special process of securing the insert to the body, employing an exceptionally strong bonding metal, and you know why a Coromant 776 bit lasts longer. In 1955, one billion feet were drilled with these inserts, all fitted to Sandvik Coromant bits or integral steels. *Nothing is more conclusive of the quality of Coromant bits than this figure.*



LOW QUALITY TUNGSTEN CARBIDE

These are unretouched, 1200-times enlarged microphotos. Above, carbide full of impurities. Those black marks are contaminations which are present when production and quality control are deficient. Contamination of this kind weakens the carbide and reduces its working life.



SANDVIK COROMANT TUNGSTEN CARBIDE

This is Coromant carbide. Notice the uniformity of size and the even distribution of grain. Coromant inserts are free of dangerous porosity and impurities—the reason they go further, have greater strength.

Nothing stands the strain like the Swedish body of a Sandvik Coromant bit

When you put the strongest possible tungsten carbide into a rock bit, the body has to be the strongest available to take the extra strain. That's why Coromant bodies are made of high-quality Swedish alloy steel. But that's not all. Inserts and clearance are cylindrically-ground and the insert ends precision-tooled to exactly the same height. This means *smoother* drilling and *smoother* holes, because the load is equally distributed on all four inserts. *Precision engineering such as this give Coromant bits a longer life!*

Nothing fits like the precision-milled threads of a Sandvik Coromant bit

In order to get a smooth profile of the highest accuracy, Coromant threads are precision-milled in a special thread-milling machine and not made with a tap. Precision-milling too protects the skirt from common fatigue failures.

SANDVIK COROMANT 776 BITS

and Sandvik Coromant integral steels are available in standard sizes through Atlas Copco, who, in their own field, are the world's largest manufacturers of rock drills. Contact any of these offices *today* for further information and a demonstration.

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CANADA, Atlas Copco Canada Ltd., Montreal, Airport, P.Q.
MEXICO, Atlas Copco Mexicana S.A., Apartado Postal 56, Torreon, Coahuila.

Atlas Copco

Manufacturers of Stationary and Portable Compressors, Rock-Drilling Equipment, Loaders, Pneumatic Tools and Paint-Spraying Equipment

FOR some years defense requirements have continued to aggravate a difficult nickel supply situation. In order to maintain civilian supplies and at the same time provide sufficient metal for defense, the U. S. government in 1952 reopened the World War II Nicaro, Cuba nickel plant. It is now operated for the government by Nickel Processing Corp., a subsidiary of National Lead Co. In the fiscal year ending June 30th the plant produced more than 31 million lb, and, with an additional investment of $43 million, the capacity will reach 49 million lb in a few years. Nickel oxide from Nicaro is refined at the only commercial nickel refinery in the U. S., the Crum Lynne, Pa. plant of National Lead Co., which commenced operation this June.

Incentive for expanding nickel capacity has come from provisions for fast amortization, as well as government contracts to buy the metal at prevailing market prices, and at premium prices from companies expanding production.

Freeport Sulphur is studying a sulfuric acid leach for its laterite ores in Moa Bay, Cuba which it plans to refine in the U. S. Bethlehem Steel Co. is also experimenting with leaching techniques on its Mayari, Cuba ores. National Lead Co. is attempting to recover nickel and cobalt from a complex sulfide ore at Fredericktown, Mo. And a silicate deposit at Riddle, Ore. is being developed by a subsidiary of M. A. Hanna Co. The U. S. government has also concluded contracts with major Canadian producers, but supplies are not yet adequate and additional premium price contracts are being pushed. By these means the U. S. hopes to increase nickel supplies from some 300 million lb at present to 440 million lb by 1961.

U. S. government purchases for the strategic stockpile of nickel, which has reached the halfway mark of its objective, are being diverted this year on a greater scale than ever before. In 1955 almost 30 million lb were so diverted, while during 1956 more than 74 million lb are scheduled to be allotted to civilian uses. Thus, both the short and long range offers some slight hope of an improved nickel supply outlook.

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THE American Mining Congress last month again urged broad government policy designed to provide protection for domestic mineral producers. Resolutions taken on certain items of policy are here reported through excerpt or summary:

Government Reorganization—AMC favors appraisal and evaluation of executive departments and agencies to eliminate competing and overlapping responsibilities and to achieve economy and efficiency.

Geological Survey, Bureau of Mines, Bureau of Land Management—Further efforts are urged to transfer all federal responsibilities affecting mineral resources to the Department of the Interior. Continued efforts to furnish improved service to the public are commended and support is given to adequate appropriations for geologic and topographic mapping.

Solid Fuels—AMC advocates sound national policy to maintain solid fuel industry growth at a level insuring sufficient supply in peace and war. Increasing use and dependence upon coal in expanding power industry is recognized.

Labor Relations—AMC believes that compulsory unionism is wholly incompatible with the principle of individual freedom—favors continued recognition by the federal government of the right of the states to legislate on the subject of compulsory unionism, and opposes the enactment of federal legislation which would deny that right—urges protection of the economy against uncurbed labor monopoly, and enforcement of law prohibiting the use of union funds for political purposes.

Social Security—Taxes collected for the support of social security should be placed in a trust fund for the payment of benefits provided for by the act.

Mine Safety—AMC considers providing safe and healthy work conditions the primary responsibility of the mining industry. A positive program is endorsed, including the following policies: participation by operators in regional meetings to discuss problems and exchange ideas; expansion of medical research programs by industry; stimulation and support of activities of safety education programs. The Bureau of Mines should receive adequate funds to continue and expand its activities in developing and disseminating improved techniques in accident prevention and promoting safety education.

Mine Financing—Reasonable requirements to avoid misrepresentation or other abuses in raising funds for mining ventures are advocated, but they should not be arbitrary nor unduly restrict honest efforts to obtain risk capital. The Small Business Administration should liberalize its qualifications for granting loans for worthy mining enterprises.

Government Expenditures—AMC supports the principle that federal governmental activity should not extend to matters which private enterprise or local or state agencies are able to carry out.

Uranium—Contracts for construction of new uranium mills should not jeopardize the investment in, nor duplicate, existing facilities. The Atomic Energy Commission should give consideration to the extension of existing and proposed mill contracts beyond 1962 in order to justify the substantial investment required and to facilitate long-term financing.

Radio Frequency Allocations—AMC urges the Federal Communications Commission to allocate additional frequencies for use in the mining industry. Most of the over 8000 radio stations in use by mine operators are using five channels shared with unrelated activities such as farming and ranching.

Water and Air Pollution—Direct federal aid in treatment plants will tend to delay rather than encourage needed construction. Congress is urged to provide accelerated amortization for pollution control facility installations.

Public Land Policy—Except in cases of clear necessity, withholding public domain lands from mining location, either through creation of new

withdrawals or maintenance of existing withdrawals, is opposed. Recommended is an amendment to the General Mining Laws which will afford, prior to discovery, reasonable protection to those engaged in exploratory work.

Gold, Silver, and Monetary Policy—AMC recommends: abolition of restrictions on purchases, ownership, and sale of gold and silver by U. S. citizens; Treasury purchase at the monetary price of all newly mined domestic gold and silver tendered by producers; use of gold and silver for monetary purposes only.

Tariffs—AMC favors enactment of excise taxes or flexible tariffs on imports, which may be suspended whenever prices are at an economic level that will permit the domestic mining industry to maintain an adequate *mobilization base* for national security. It disfavors placing power to regulate U. S. trade in the hands of any international body. Stockpiling is endorsed; minerals should be processed to usable form while capacity is readily available.

Taxation—Tax system must be such as to yield needed revenues without discouraging the investment, risk, and effort necessary for income-producing activities from which the revenues should flow. In no case should the over-all rate on income of the individual or corporation exceed 50 pct and in due course should be reduced to not more than 35 pct. Exploration expenditures should be fully deductible. New mines should be exempt from taxation for three years after the beginning of profitable operations.

◆◆◆◆◆◆

WHEN one of the little uranium ventures settles down to work its claims, the biggest problem invariably turns out to be money. Money is the salve to the long, hair-tearing period between discovery of the ore body and profitable production. For the small firms, keeping the treasury plump enough to maintain exploratory drilling operations has sometimes meant mortgaging equipment and more than one of the little operators have bemoaned the fact that their issue of shares brought in less cash than was needed. Meanwhile, long-established big companies can pay for their drilling with income derived from other business. When finally the hole-sinking does prove a good lode, if it does, the small firm often finds that there is little money left for the process of carrying out the ore.

One swing has been toward making arrangements with large concerns to do necessary drilling on a percentage basis. In general the property, in this case, must be a very likely prospect. When it is done, the small firm can use its capital for hiring skilled people and packing the ore off to a mill. If the ore body is a marginal one, the little outfit may have to halt production when the first 10,000 lb of concentrates are turned out and the AEC bonus comes to an end.

One thing in small ventures for uranium is hard to deny, if the mere jeep-and-geiger-counter crews are not discounted. More money is spent in tracking down the elusive ore than is derived from it. Exceptions are not the rule but the good strikes make the failures seem worthwhile.

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INDUSTRIAL research has come a long way from the days of the woodshed or kitchen-sink laboratory. Extensive facilities, special equipment, luxury, even glamour have become prominent ingredients of the laboratory installations that highlight the modern industrial research program. The basis is still the people who come up with ideas, but there is much emphasis on making the surroundings as favorable as possible to attract and hold these people with ideas.

As research grows by leaps and bounds, there is a tendency to separate levels of activity. Labels vary —fundamental, basic, applied, developmental, product—but the pattern is similar: a shifting of company-wide basic research activity to one permanent installation located away from works and plants. At the same time, applied research and development is put in closer contact with production.

Suburban sites fit the pattern of these centralized basic or fundamental research laboratories, but sometimes the community itself has strong reservations about the prospects of embracing an industrial neighbor. The *Cleveland Plain Dealer* took note of this problem in an editorial:

"The best answer to the resistance to research laboratories which still pops up in some suburban communities is a close look at a handsome new one like Union Carbide's in Parma (Ohio.)

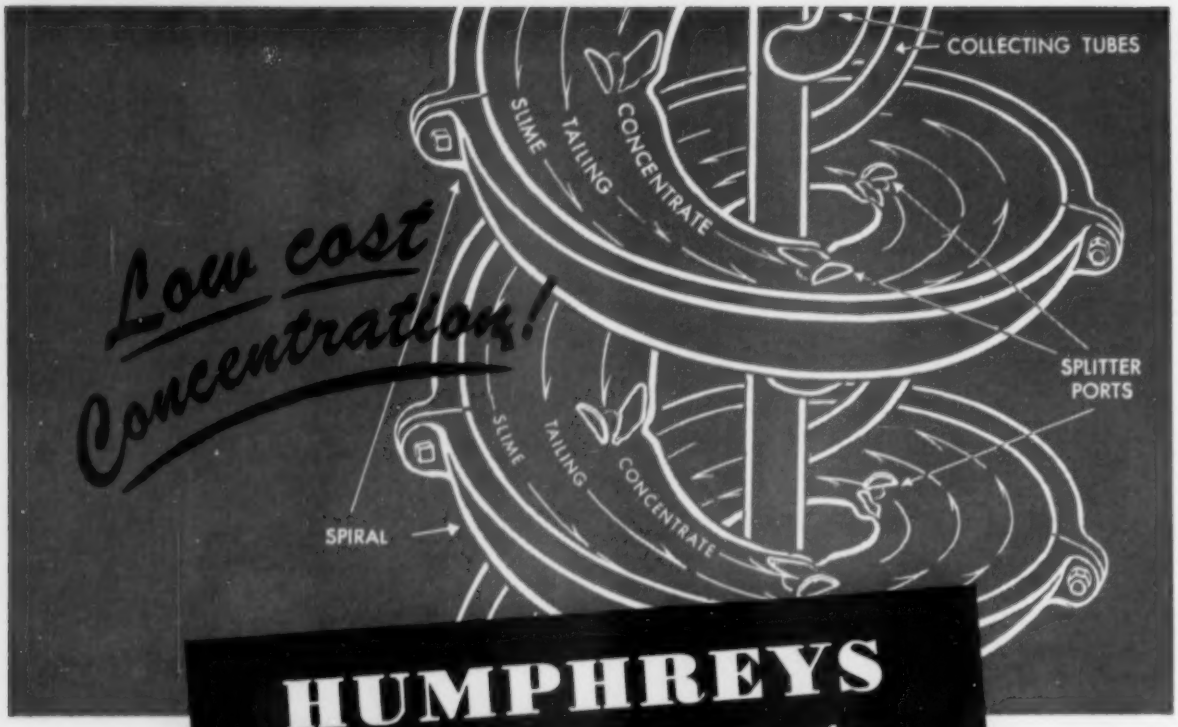
"Thus, Carbide officials could perform a unique public service by opening their doors to groups of citizens and officials from communities where laboratories become an issue.

". . . The new Carbide (actually a UC&C subsidiary, National Carbon Co.) lab is a beauty. Both inside and out, it is far more like a university than an industrial center. The landscaping and the modern new building are in the pattern of a fine campus. And inside, too, the libraries, the research equipment, and even the scientists, lend a pleasant and impressive academic tone.

"One look at this center should be enough to convince anyone that a research laboratory is an excellent neighbor."

A pattern has developed for the architectural handling of the buildings generally. Often the structure has the administration and reception sections forming a core, and located up front. The working research area extending back from this basic unit is usually given provision for further expansion to the rear.

But buildings and sites remain only outward manifestations of the more important changes taking place as industry joins government and the universities as a sponsor of research. J. A. Krumhansl of National Carbon puts it this way, "If there is a single prominent feature of the present vigorous expansion of industrial research that strikes one, it is the broadening of *scope* as well as increase in volume which is taking place."



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Throughout the world economy-minded mineral producers rely on Humphreys Spirals . . for low-cost concentration and increased production. These highly efficient concentrators are noted for economical installation, low-cost maintenance and dollar-saving operation . . . plus trouble-free performance over long periods of operation. No moving parts. Require very little floor space.

APPLICATIONS:

- Production of a concentrate and a tailing in a single rougher stage.
- Production of a bulk concentrate of several heavy minerals and a finished tailing.
- Scavenging the tailing from another process for the recovery of heavy minerals not amenable to the primary processes.
- Production of a finished concentrate, with a tailing to be treated by another process.

Write today for information on metallurgical tests of your ore samples for spiral treatment.

Engineering Division

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Meetings for Maximum Member Service

The Meetings:

WE had some first-hand evidence of the vitality of meetings at the local, regional, and divisional level in the past few weeks. It was our good fortune to attend several meetings, our misfortune to miss others. Two meetings conflicted for us, not for the people they aimed to serve, and that is perhaps a most important point to keep in mind when thinking of membership service. To the few men who have an opportunity to travel across the country, a mining meeting in Michigan and a metals meeting in Ohio can represent a conflict. To the members in those areas they no more conflicted than either one did with a meeting in Colorado, and there was one the same week.

This issue carries a report on one meeting, the well-established Rocky Mountain Minerals Conference. And, at the expense of other worthy ones, we would like to mention three outstanding local meetings which fell in the period of late September, early October.

In the Northeast, the Adirondack Section swung over to Vermont for a visit to Appalachian Sulphides and held their dinner get-together at South Stratford, Vt. That same weekend the Black Hills Section put on a real display of logistics with a luncheon at Edgemont, S.D., followed by a smoothly scheduled afternoon of field trips to a new uranium mill and two producing mines in the area. The section carvalcade then swung back toward Rapid City for a dinner at Sylvan Lake before dispersing to homes scattered over most of the southwest corner of the state. (The next week and a little farther west found the Utah Section busy with its sponsorship of the Rocky Mountain Minerals Conference.) At this point a jump north and east to the Upper Peninsula Section where a mine-to-mill-to-dockside look at the U.P.'s newest iron ore beneficiation operation was followed by a reception and banquet. A more complete report on these three meetings will appear in AIME News, in the December issue of *ME*.

During this same period—late September, early October—a quick glance at activities primarily of interest to members of the Petroleum or Metals Branches would find gatherings scheduled by three sections of the National Open Hearth Conference. The NOHC Southwestern Section put on a two-day annual meeting and field trip in Colorado, the Southern Ohio Section was busy at Columbus, and the Eastern Section met at Philadelphia to see the growing shape of Eastern Seaboard steel industry. Right in the middle of this same time-span the IMD scheduled its three-day annual fall meeting in Cleveland during Metals Show week. And, out on the West Coast the Petroleum Branch was dusting off the chairs and readying the red carpet for its big fall meeting.

This is not a comprehensive listing of September-October AIME meeting activity, it just includes meetings we noted in planning a two-week swing across the country during that period.

As the curtain was going down on these meetings a new list was coming up. In the East two meetings of particular Mining Branch interest: the Northeastern Mining Branch meeting at Hershey, Pa., and the AIME-ASME Joint Solid Fuels Conference at Washington, D.C.

Cover all these meetings? It would run three men ragged. In fact the multiplicity of meetings does put a burden on the hypothetical member with unlimited time and funds for travel who wants to attend *all* worthwhile local, divisional, and regional meetings of the Institute. Our AIME Presidents are faced with this problem to a degree, and it does put a tremendous burden on their time. But, it does mean more service to more members.

The Pattern:

A pattern is taking shape—taking the shape of things to come in AIME meetings aimed for maximum membership service. There is the promise of sound growth in the meetings mentioned above because they were self-generated—came about in response to membership needs. If any meeting, convention or show fills a need it will not only be a success, it will be a continuing success. The people to make that kind of meeting continue will come forward. And in that idea perhaps lies a summary of the growing pattern of AIME program activity—service to members made possible by the service of those members sponsoring the meeting. If the need is genuine and the need is met, the audience is assured and the playrights, producers, and stagehands will come forward.

In this developing pattern there are, first, the more expansive local section meetings covering a considerable area technically or geographically. To meet the needs of larger areas there is the outline of a pattern of regional meetings covering the Northwest, Southwest, Rocky Mountain, Northeast, and Southeast regions. To meet the needs of the specialist there are gatherings that feature technology in depth. Finally there is the Annual Meeting, whatever its format. Gaps still show up, but the picture of the pattern to come is fast being shaded in.

Somewhere there is a point of diminishing returns in the variety and number of technical meetings. Your viewpoint of where this point lies depends somewhat upon your financial resources and time available for travel, as well as upon your breadth of interest. But, it is undoubtedly better to have an embarrassment of riches than the other kind.

R. A. B.

How Bucyrus-Erie Bonus Quality Helps You Cut Costs, Increase Output

This is the Bucyrus-Erie 38-B shovel that won praise from its owner in this stripping operation near Oliver Springs, Tenn.

To discover the real advantages of Bucyrus-Erie *Bonus Quality* put a Bucyrus-Erie shovel to work in your pit. That's what Chas. Walls and Sons, Oliver Springs, Tennessee did.

Walls' job was to strip four feet of sandstone and 25 feet of shale and dirt to get at the coal in a mine near Oliver Springs, Tenn. The complete job involves excavation of 1,200,000 yards of overburden. After putting a Bucyrus-Erie 1½-yd. 38-B to work for a year, Walls commented, "Excellent service, strongest built of any shovel of its rated capacity."

"Strongest built" is just a part of the advantages of *Bonus Quality*. Not only do you get design and construction that adds years of life and keeps maintenance costs low, but there's extra performance value, too. There is high-speed operation that delivers big output. There is dependability that *keeps* big output coming day after day.

When you plan to add a shovel to your operations, be sure to get the story of *Bonus Quality* from your Bucyrus-Erie Distributor. Remember to see him soon. 243E56

Bonus Quality Bucyrus-Erie uses highest quality steel, then flame hardens it to bring you gears and pinions that wear long, keep costs low.

BUCYRUS ERIE

South Milwaukee, Wisconsin

High Speed Photography Used to Redesign Conveyor Transfer Point

by D. J. Reed, Jr.

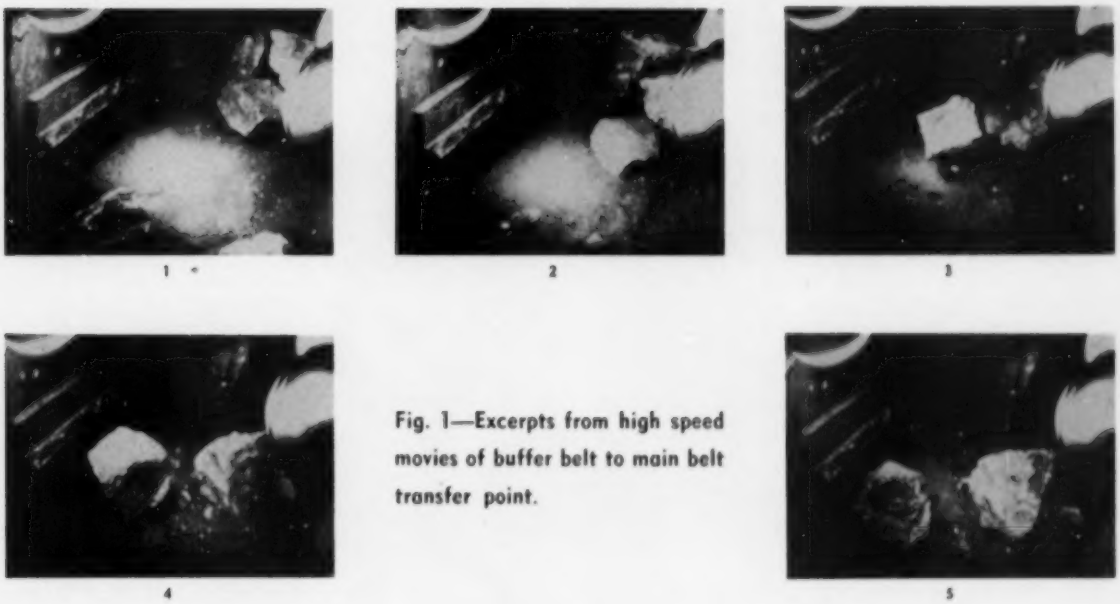

Fig. 1—Excerpts from high speed movies of buffer belt to main belt transfer point.

A solution for redesigning the main belt to buffer belt transfer point.

CONCORD coal mine near Bessemer, Ala., built, owned, and operated by Tennessee Coal & Iron Div., U.S. Steel Corp., produces only a metallurgical grade for use as coke in blast furnaces of the division at the Fairfield and Ensley works. Capacity of the operation is approximately 800 tph, half of which is delivered to the coke and coal chemical plant at Fairfield after beneficiation to metallurgical grade coal. The remaining portion of the mine run is rock and refuse removed during preparation.

Concord mine coal is hoisted to the surface coal preparation plant by an underground slope conveyor. The slope conveyor is 2449 ft long and has a lift of 729 ft. At time of installation the Concord mine slope conveyor was the largest single conveyor unit ever made with respect to belt horsepower and total lift. After several years it became necessary to determine the cause of slope conveyor cover wear.

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Main line haulage locomotives drop off their trips in a double-tracked load yard above the rotary dump. Here a series of car hauls picks up the trips and feeds them to a car haul ahead of the dump. This car haul automatically feeds and centers two cars for dumping in the rotary dump. The trips remain unbroken as they pass through the full revolving dump. Empty cars run by gravity below the dump, over car retarders, into the double-tracked empty yard. Locomotives come into the empty yard on a runaround track to pick up their cars for a return trip to sectional sidetracks.

The mined coal and rock falls into a hopper below the dump from which it is fed, by parallel lines of apron feeders, to a stationary grizzly and single roll crushers. The grizzly undersize and the crushed oversize drop into a double surge bin. From this bin two apron feeders load the mined material onto a short center *buffer* conveyor belt, which in turn loads the material onto the main slope conveyor.

Wear on the main slope belt is kept at a minimum by maintaining equal speed between the material sliding on the belt and the main belt itself. This is accomplished by running the buffer belt at the same speed as the main belt, by maintaining a minimum drop from the buffer belt to the main slope belt, and by proper transfer chute design. High speed camera studies of the transfer point, however, have proved the inefficiency of operating the buffer belt at the same speed as the main belt.

The first underground high speed photographs in the Concord mine were made during May 1953. The purpose of this study was to photograph the buffer belt to the main belt transfer point to determine whether large material penetrated into the main belt carcass. These movies, Fig. 1, were graphically descriptive of material transfer, but conclusive results were not obtained. Limitations of camera lens field width, inability to photograph material transfer at an angle normal to conveyor centerline and parallel with the belts, and interference with normal mine operation all made it necessary to simulate the transfer point by conveyor prototypes outside the mine.

The high speed movie of Concord buffer belt to main belt transfer point showed rock impact on the main belt with a reverse or backward rotation after the impact. Material transferred from the buffer belt was given a rotational force by the main belt as the material bounced into the air and as the main belt moved from under the particle. Resultant effective forces on the material generally did not lie in the plane of rotation of the material center of gravity, thus causing the material to bounce over the belt width. Primarily, the movies showed scuffing action on the main belt cover. It was desired to observe and measure empirically the belt wear during material transfer and rock penetration into the main belt carcass. Of the utmost importance, however, as revealed by the camera, was the redesign of the transfer point to effect a smooth material transfer.

Tests formulated to utilize various parameters for measuring material transfer were all lacking in one respect—they could be applied on the underground transfer station only with difficulty. The solution would have been to photograph the transfer point from any angle in a circle slightly above the plane of the main belt, except for inaccessibility and obscured vision at heavy loads. Obviously, the best location for the movies would have been 90° to the longitudinal axis of the belt, but this point is obscured on both sides by loading skirts and baffle plates. The disadvantages of this location were the conveyor down time incurred by removing skirts and baffle plates, and the hazards of rocks bouncing off the main belt. It was therefore agreed to build the prototype conveyors at the Concord mine construction warehouse.

The conveyors and structures were designed and built during August and September 1953. One 33-ft conveyor was the material accelerating conveyor or buffer belt prototype, and a 19-ft conveyor was the material receiving conveyor or main belt prototype.

The first series of tests was run in nine parts on the theory that material should be transferred from the buffer belt with speed identical to that of the main belt, although it was not necessary that conveyor speeds be identical. The bottom of the buffer belt 30-in. head pulley was placed 6 in. from the main belt. It was computed that a buffer belt speed of 815 fpm would transfer material with a velocity component parallel to the main belt identical to the main belt velocity of 580 fpm. The first nine tests were completed by running the buffer belt from 575 to 850 fpm in increments of 50 fpm without a transfer chute.

Results clearly indicated that the vertical velocity, acquired by the particles in the 36-in. trajectory, was great enough to cause extreme material bouncing on the conveyor belt. Also, the theory held by many that a belt loading point should have very light tension was proved to be, in this case, a fallacy. The movies showed material depressing the main belt, and by resulting belt tension the material was thrown 2 to 6 in. back into the air.

The second series of tests was conducted in November 1953 with a 3-ft belt separation with a flat chute on 25° slope. With high buffer belt speeds, test results were better than the first series of tests without transfer chutes but left a great deal to be desired. Material transfer was very poor with respect to reverse rotation of particles but little initial bouncing was noticed.

Three tests were conducted with 634, 700, and 800-fpm buffer belt speeds and 580-fpm main belt speed.

The third series of tests was conducted during February 1954 after the conveyors had been modified and a chute installed between the conveyors. Belt separation was increased from 3 ft to 4 ft 8 in. to match the underground installation. A troughed belt slide was constructed and placed under the main belt to entirely eliminate idler characteristics. A 6x7-ft graph board was placed behind the conveyors to show exact motion of particles before and after impact with the main belt.

Three tests were made with a flat 30° slope transfer chute at buffer belt speeds of 634, 700, and 800 fpm, with the main belt speed 580 fpm. These tests showed that material transfer, even with the transfer chute, was not satisfactory at the high buffer belt speeds. Also, the transfer chute end was above the horizontal axis of the buffer belt head pulley with the attendant difficulties of stoppage of material between the buffer belt head pulley and the transfer chute end. Any rocks or steel wedged between these two points severely cuts the cover of the buffer belt. The transfer chute location was one difficulty that had to be corrected in the final design.

Two additional tests, making a total of 17 tests, were made without a transfer chute to photograph trajectories of material leaving the buffer belt head pulley. The purpose of these films, Fig. 2, was to compare actual particle trajectories to computed particle trajectories used in conveyor design. Test results showed that identical particles exhibit similar but not identical performance. In a vertical drop of 4 ft 8 in., at 634-fpm buffer belt speed, a 4-in. horizontal distance variance was observed. In a later test, this same behavior was exhibited in transferring material to a hinged chute. The chute was used with a high buffer belt speed and, of necessity, the end of the chute was located several feet from the buffer belt head pulley. For the chute to have sufficient angle to move the material and to be of sufficient length to receive the entire thickness or nap of material, the chute rear end location was critical. The trajectory of material adjacent to the buffer belt cover was designed to clear the chute end by 1 in. However, approximately 5 to 10 pct of the material hit the chute end. All the material

Excerpts from high speed movies taken at transfer point.

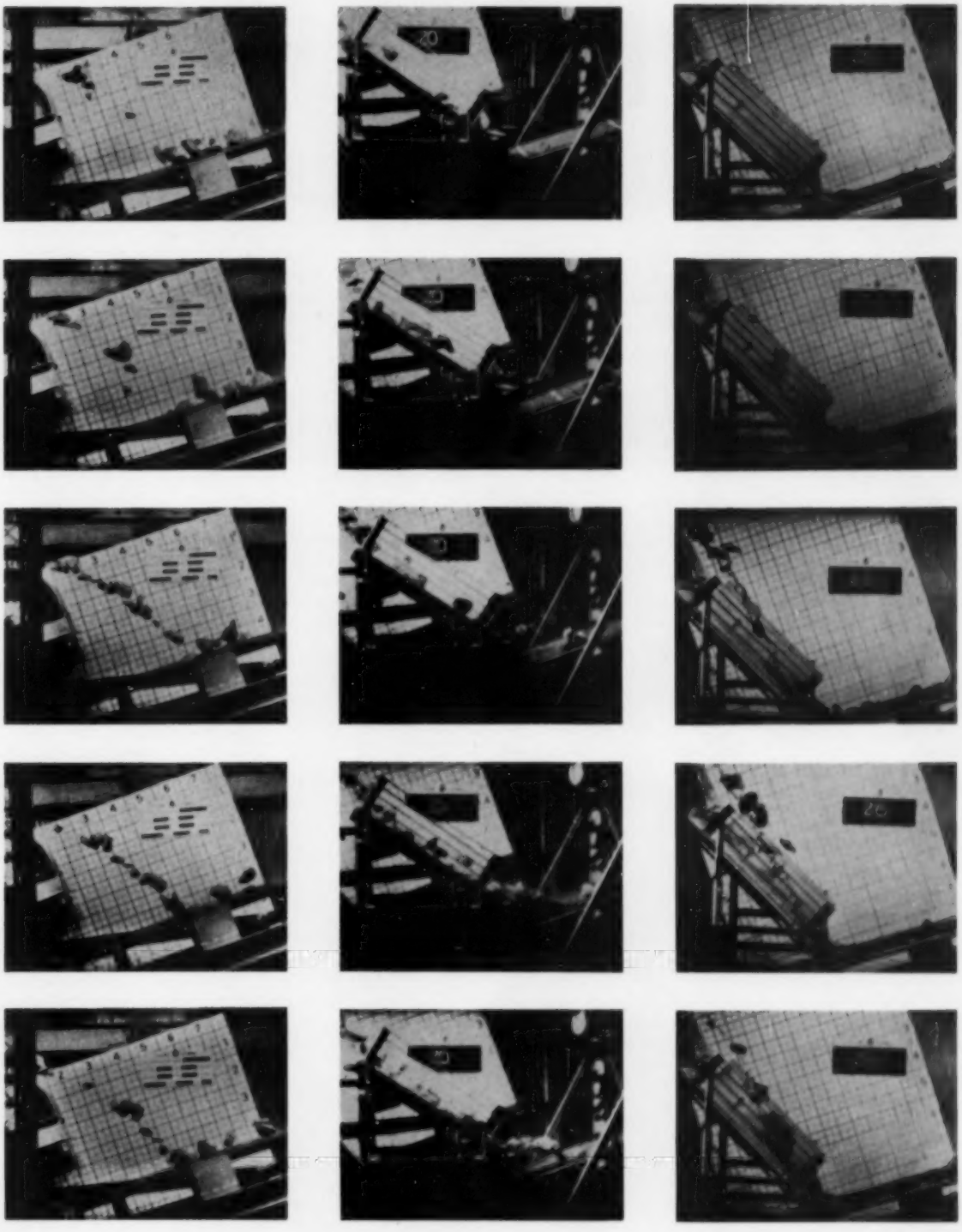


Fig. 2—Trajectories of transferred material.

Fig. 3—Material transfer over a double chute.

Fig. 4—Material transfer over a single chute.

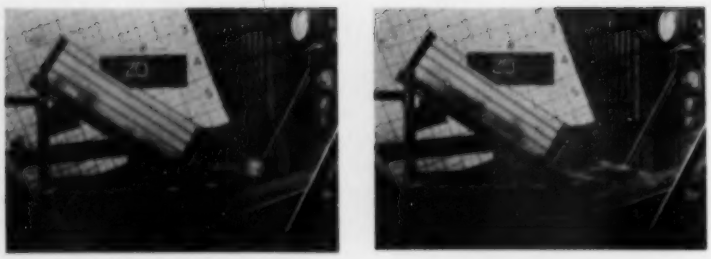


Fig. 5—Excerpts from normal speed movies of material transfer over a double chute.

transferred in these tests was −5-in. +2-in. sandstone and shale. The difference in trajectories of these particles was assumed to be caused by air resistance.

Subsequent to the third series of films, a detailed study was made to analyze all variables and incorporate the opinions of all concerned with the problem. By March 1954 major conclusions had been reached concerning the most effective point for material transfer:

The buffer to main belt separation necessarily would have to be increased from 4 ft 8 in. to at least 7 ft 6 in. To accomplish a greater belt separation, without increasing the mine roof height at the transfer station, it would be necessary to shorten the buffer belt and increase the belt slope from 17° 21′ to 21° 5′.

With a buffer belt speed of 634 fpm, the only practical alternative was to have a double chute composed of a dribble and a main chute. The dribble chute would catch material falling off the buffer belt during the time the buffer belt is accelerating from zero to full speed. The main chute would transfer material after the buffer belt is at rated speed.

With slower buffer belt speeds, a single chute could be utilized. For proper design, the chute would be below and behind the foremost part of the buffer belt head pulley to catch dribble from the buffer belt when started up with material on the conveyor. The existing buffer belt width determined the slowest speed at which the required tonnage could be conveyed without exceeding designed belt capacity. Transfer chute slopes and mine roof headroom were other factors considered in the design.

It was observed from high speed movies that flat chutes allowed material to gain considerable velocity perpendicular to the conveyor belt causing material bouncing and belt cover wear. Curved ends were incorporated in the design of test chutes to deflect the material into the conveyor belt direction of travel rather than leave this function to the conveyor belt only.

In the fourth series of movies six films were made with a dribble and main chute assembly, a double chute. Buffer belt speed was 634 fpm and main belt speed 508 fpm, with coarse material and with coarse and fine material. Material transfer was improved, but the films, see Fig. 3, did not show the desired results. The material still bounced slightly and rotated backward before settling down on the main belt. Two of the movies were made by a normal speed camera for purposes of comparison with the high speed movies.

One difficulty that developed early in the project was that many could not conceive the extremely short interval of time required for high speed movies. The normal speed movie, Fig. 5, shows the blurred effect seen by an observer during material transfer.

The fifth series of movies consisted of three films made of a single chute assembly, with buffer belt speed of 467 fpm and main belt speed of 580 fpm, using coarse material. Buffer belt speed was reduced from 634 fpm as in the actual underground installation. Test results, Fig. 4, showed the material sliding on the transfer chute before being turned in the direction of the main belt by the curved chute end. Material, as it left the end of the chute, had a slight forward rotation while moving through the air before striking the belt. A force was applied to the bottom of material as it impinged on the main belt, causing a rotation about the particle centroid. If the particle bounces off the belt to an appreciable height, the rotation aggravates the initial disturbance considerably. However, it appeared that the forward rotation of material leaving the chute was arrested after initial impact on the main belt. The result was that material landed on the belt and with practically no motion. All particle motion was arrested within 1 to 3 ft of the chute end.

The chute and buffer belt speed combination appeared to be an excellent solution to the problem of redesigning the main belt to buffer belt transfer point.

Tests 27 and 28, made in December 1954, were movies of wet coal fines being transferred over main belt and buffer belt prototypes. Results showed that fine material could be handled as well over the transfer chute as coarse material.

The conclusions reached during the course of experimentally redesigning the Concord mine buffer belt to main belt transfer point are as follows:

1) Buffer belt speed must be decreased from 634 fpm to 467 fpm to diminish trajectory of material leaving the buffer belt head pulley.

2) Separation between the main belt and buffer belt must be increased from 4 ft 8 in. to 7 ft 7 in. for installation of an effective material transfer chute.

3) The buffer belt slope must be increased from 17° 21′ to 21° 5′ by a vertical curve in the conveyor immediately after the discharge point of the last material feeder. The increase of buffer belt slope is necessary to obtain headroom required for a transfer chute.

An effective transfer chute is defined in this case as a single chute 1) located behind and below the forward most part of the buffer belt head pulley, 2) having sufficient slope to move the material, and 3) having provisions for changing direction of transferred material from perpendicular to parallel to the main belt conveyor. The location specified is necessary to prevent wedging of material between the buffer belt head pulley and the transfer chute upper end. Material between these points cuts the buffer belt cover. The curved chute that changes direction of the material will contribute substantially to reduced wear of the main belt cover.

mately 1.20 As:1.00 Fe. This ratio is an important factor in achieving optimum chemistry in the oxidation autoclave.

encountered in mechanical operation of the oxidation autoclave. Gasket and valve failures were fre-

Pressure Leaching and Reduction at the Garfield Refinery

by J. S. Mitchell

Cobalt recovery from arsenical concentrates by pressure leaching and reduction involves special methods and operating problems. Principal steps in the process are auto-oxidation acid leaching under pressure, filtration of the tailings, purification of the solution, hydrogen reduction of the ammoniacal solution, and electrical furnacing for sulfur removal and granulating the metal.

EXTRACTING cobalt metal from arsenical ores has presented a major metallurgical problem for many years. In acquiring the Blackbird mine near Salmon City, Idaho in 1943 Calera Mining Co. accepted the challenge to solve this problem.

Originally much experimentation and piloting had been carried out on a process involving two or three stages of roasting, followed by leaching and chemical separation of cobalt metal from the leach liquors. However, before plant design had been started, Calera became interested in a new process sponsored by the Chemical Construction Corp. which appeared to offer considerable simplication, lower operating costs, and better recovery. It was finally decided to adopt this scheme, in spite of the fact that it had not been piloted.

For various reasons, which need not be discussed here, developing the process presented more difficulties than were anticipated. Calera struggled with this situation for more than a year, solving many of the more serious problems and producing a substantial amount of metal, but it also became obvious that more equipment would have to be installed and a much longer period of costly mechanical and process development would be required before a commercial rate of production could be expected. Considering these circumstances, Chemico agreed to undertake responsibility for this continued development, and during their period of operation (nearly two years) real progress was made. By the end of this time it had been demonstrated that an agreed production rate could be obtained, with acceptable recovery and cost factors. In December 1955 Calera reassumed operation of the plant, and since that date it has operated satisfactorily. Efforts to improve processing and costs are continuing. Costs, especially, can be lowered considerably, partly through continued mechanical improvements which, it is hoped will greatly lower maintenance costs on the high pressure equipment. Various other possibilities for improving metallurgy and lowering costs are being diligently explored.

The complex beneficiation of the ore at the mine produces a cobalt concentrate that is treated at the company's refinery near Garfield, Utah, and a copper concentrate that is shipped to custom smelters.

This article will deal with the general methods employed and the operating problems encountered in developing the high pressure leaching and subsequent treatment of the cobalt concentrate carried out at the Garfield refinery.

Concentrate: The concentrate arrives at the refinery in railroad boxcars in approximately 50-ton lots. The material is removed from the cars with mechanical loaders and temporarily stored in individual bins pending subsequent blending and introduction to the plant system. Each carload lot maintains its identity up to the time it is charged to the concentrate mix tank.

Analysis of Concentrate Arriving at Garfield Refinery, Percents

Cobalt	17.50
Iron	20.00
Arsenic	24.00
Sulfur	29.00
Nickel	1.00
Copper	0.50
Insoluble	5.00
Moisture	12.00

On the basis of arsenic and iron content the concentrate is blended to achieve a ratio of approxi-

J. S. MITCHELL is Plant Manager at the Garfield Refinery of Calera Mining Co., Garfield, Utah.

these cements have been retreated, yielding the copper and cobalt as separate products.

produce complete conversion of the cobalt content of the feed solutions to metallic powder presented

mately 1.20 As:1.00 Fe. This ratio is an important factor in achieving optimum chemistry in the oxidation autoclave.

The concentrate is repulped with process water or dilute process liquors to a pulp density of 1250 to 1300 or approximately 25 to 27 pct solids.

For all practical purposes the subsequent handling of this slurry is achieved with conventional agitation and pumping equipment. When slurry is transferred from the initial repulping tanks to storage feed tanks the material is passed over a vibrating screen for removal of debris and oversize that may interfere with subsequent pumping.

Up to this point, no serious operating problems have been encountered. Use of acidic make-up solutions requires rubber-covered or stainless steel equipment.

Oxidation: This operation is carried out in a horizontal pressure vessel 40 ft long and 6 ft in diam. The shell is of carbon steel with 3/16-in. cladding of type 316 stainless steel. In addition to the cladding, the vessel has a 12-lb lead lining and acid brick.

The lead lining is an additional safeguarding feature, installed at the time the first acid brick lining was replaced after approximately three years of service. Inspection of the stainless clad after the failure of the initial acid brick lining revealed that in spite of the precautions taken, the clad had been attacked to a considerable degree.

The autoclave is divided into six equal compartments by acid brick walls with overflow weirs, which allow the oxidation slurry to flow from one compartment to the other. Each compartment contains an agitator mechanism to provide mixing and aeration. Provisions are also made to sparge air into each compartment.

Concentrate slurry is introduced to the first compartment by means of Manton-Gaulin high pressure positive displacement pumps. The original pumps were of the triplex plunger type. Severe abrasion and corrosion resulted in extremely high maintenance costs; consequently these units were replaced with Mantan-Gaulin Hydrex pumps. These newly developed hydraulic displacement pumps are giving acceptable service at this time.

Addition of concentrate, process solutions, and recycled process slurries and the discharge of the oxidized slurry is a continuous operation. Operating level is controlled by a Gagetron.

The chemical reaction is exothermic, producing an oxidation slurry containing free acid, cobalt, nickel, iron, and copper as sulfates and minor amounts of arsenic in solution, also iron-arsenate and calcium sulfate as an insoluble residue. Operating pressures are maintained at over 500 psig and temperatures in the various compartments are over 375°F.

Exit gas from the autoclave (oxygen deficient air and steam) is exhausted through scrubber towers to the atmosphere. Oxygen utilization from the compressed air introduced to the reactor approximates 60 pct.

Oxidation process solubilizes 95 to 97 pct of the cobalt. However, there are small losses of soluble cobalt in the subsequent filtration step.

To the uninitiated the process explained thus far seems quite straightforward. As far as flow of material is concerned, this is true. The problem of containing the process was another matter. During early stages of development many difficulties were encountered in mechanical operation of the oxidation autoclave. Gasket and valve failures were frequent, and packing life of the agitator shafts was of short duration. Prior to acquiring titanium bolts and fasteners (at the time under development, with slow delivery) the agitator impellers fell off at the rate of one per week. At this point it might be mentioned that corrosion resistance of even the best stainless steels against the hot slurry is almost nil.

Gas let-down from the vessel (operating pressure to atmospheric) took its toll in various alloy valves. Stainless steel valves, fittings, and fastenings literally dissolved under the severe operating conditions of both erosion and corrosion, and service life was short.

After many months of *cut and try* with design and materials of construction, these items were improved to a considerable degree. In almost all instances these critical parts have had to be developed and then fabricated locally. Until recently no suitable substitute had been discovered for the titanium and ceramic that went into the slurry and gas discharge valves. The importance of titanium for agitators and discharge piping cannot be overstressed. More recently tantalum and Refrax R and silicon carbides have shown promise for these applications.

Filtration and Solution Purification: Acid slurry discharged from the oxidation autoclave is substantially neutralized by addition of limestone and hydrated lime. Some precaution is necessary at this step to avoid precipitation of cobalt hydroxide. Formation of this compound would result in its loss with the insoluble iron arsenate-gypsum residue that is discarded to waste in subsequent filtering operations.

Filtering of the oxidation slurry is achieved on conventional drum filters fitted with stainless steel drums and hard rubber decks. Polyethylene filter cloths are used on all drum filters. To reduce soluble losses to a minimum, the usual procedures of repulping, refiltering and displacement washing are carried out. Dilution of the primary filtrate is controlled to a cobalt concentration appropriate to the subsequent reduction operation.

Further purification of the process solution is necessary to remove the residual iron (that amount that was in excess of the stoichiometric requirements to form iron-arsenate in the oxidation autoclave), arsenic, and copper. The iron and arsenic are removed conventionally by addition of hydrated lime, anhydrous ammonia, and air. This process is a batch operation carried out in steam-heated rubber-lined tanks. Proper aeration and temperatures of approximately 140°F are considered essential. The above procedure produces a residue of gypsum, iron hydroxide, and minor amounts of iron-arsenate. The resulting slurry is filtered on precoated drum filters to give an iron and arsenic-free solution. The residue is recycled to the oxidation autoclave, as it contains appreciable cobalt values. The filtrate is advanced to a subsequent batch process wherein the copper is removed by simple cementation with cobalt powder.

The cementation step produced some unexpected results. To attain an acceptable time cycle for this operation it was necessary to introduce excess amounts of cobalt powder. This naturally produced a copper cement sludge that contained appreciable percentages of cobalt. This represented a *tie-up*, as well as a copper cement that was not marketable. However, a process has now been developed whereby

Idaho Almaden Mercury Mine

these cements have been retreated, yielding the copper and cobalt as separate products.

Hydrogen Reduction: The reduction step is a batch process carried out in vertical autoclaves of approximately 450-gal capacity each. These vessels are fitted with agitators, hydrogen spargers, and steam jackets.

The purified solution produced in the previously mentioned steps is now ready for the reduction process. Anhydrous ammonia is added in controlled amounts to produce a complex cobalt ammine. The resulting solution is pumped into the reduction autoclaves, and heated to about 375°F; compressed hydrogen is admitted, and in the presence of a catalyst, the cobalt is reduced to a fine metal powder, grey in color with an apparent density of 4 to 5. Total operating pressure is 750 to 800 psig.

An operating cycle consists of a *seed run* and several *densifications*. In the seed run formation of the metal powder is a result of the catalyst, which produces a fine metal powder. In the subsequent densification runs the powder from the seed run acts as the nucleus to produce powder of increased particle size and density, depending on the number of densifications.

From an operating standpoint the mechanics of the gaseous reduction phase appear to be simple. It might be said that the combination of pregnant solution, catalyst, and a controlled hydrogen atmosphere produces cobalt powder. This is all very well, but it does not account for the considerable percentage of cobalt metal that ends up as a dense metallic plating on the interior surfaces of the reduction autoclaves. Extensive research in the field of agitation, catalysts, and chemical composition of working solutions resulted in decreased amounts of this plating; however, the condition continues to exist.

When these deposits were first encountered they were removed by manually chipping with pneumatic hammers. This was costly, laborious, and time-consuming. However, this method was the only practical way that could be used at the time. Reduction capacity was limited to four autoclaves and *down time* had to be minimized. In the interim, additional autoclaves were acquired which provided increased capacity, making feasible the development of an *acid leach* for removing plated deposit.

To add to the problem, removing the deposit by chipping produced a platy product (several thousand pounds per vessel) that required considerable treatment before it could be recycled to the plant flow. A technique has now been developed whereby the material has been converted to a marketable product.

A further problem encountered in the reduction step was the production of cobalt metal powder that contained excessive sulfur impurity. This sulfur is present as entrapped gypsum particles and to some extent as contained ammonium sulfate. The latter can be removed in part by successive displacement water washes and subsequent decantation or filtration.

Initial process design had called for calcining the powder in a hydrogen atmosphere followed by compacting in a conventional pelletizing press. Since the powder did not meet sulfur specifications a further refining step became necessary before a saleable final product could be produced. This problem was solved by the installation of an arc furnace.

The fact that the gaseous reduction step did not produce complete conversion of the cobalt content of the feed solutions to metallic powder presented another unexpected problem, with accompanying loss of metal. Conversion is in the neighborhood of 97 to 98 pct. It was also discovered that some redissolution is experienced if the metal powder is allowed to remain in contact with the reduction end liquors.

Consequently reduction end liquors and wash water are now treated with ammonium or sodium sulfide to produce a cobalt sulfide precipitate, which is filtered out, and subsequently returned to the oxidation step.

Added to the above mentioned process problems were mechanical weaknesses. Agitator mechanisms were a recurring source of operating delay, partly because of repeated failure of packing glands. Discharge valves also failed frequently as a result of erosion experienced in handling the powdered metal slurry while taking a pressure drop of some 800 psi. Gasket failures were an everyday experience.

Initially the autoclaves were fitted with internal steam coils. These coils often failed at the welds, resulting in contamination of condensate and presenting further difficuties in the steam plant. Even the operation of a relatively simple steam trap became a problem when cobalt-bearing solutions found their way into the system. However, during their period of operation, Chemico replaced the heating coils with external steam jackets, which have eliminated this particular problem.

Arc Furnace: The powdered metal produced in the reduction step contains on the average 0.15 pct sulfur. This material is charged into a size *Q* 'Lectromelt furnace and melted down in the presence of a high lime slag. Minor amounts of coke are added, and some pig aluminum for degassing the metal is charged just prior to the pour.

Pours are made as soon as a sulfur analysis indicates that the metal is within specifications. By this time the bath has reached a temperature of approximately 2800°F. The molten metal is poured into a pit of water to form granules. Temperature and rate of pour have a decided effect on the physical characteristics of the final product. The granules are collected in a basket suspended in the pouring pit. Subsequently, they are dried, polished, and put up in steel containers for shipment.

Analysis of Final Products, Percents

Cobalt	96.60
Nickel	3.90
Sulfur	0.03
Iron	0.20
Arsenic	0.03
Copper	0.02
Carbon	0.15

Summary

As the first such plant of its kind to be put into production, Garfield refinery had many new and difficult problems. In a period of three and a half years many of these have been solved, while others remain to be improved.

Development work continues in an effort to simplify various phases of the process. At this time, much is being done toward replacing batch operations with continuous flow. It is firmly believed that the remaining problems related to continuity of operation can be solved in the near future, with further improvements in recovery and cost.

Idaho-Almaden Mercury Mine

Mining and Geology

by John R. Reynolds

MERCURY has long been neglected in American mining industry. The plights of domestic producers have been many and their compensations few. The mercury market has been dominated in the past, as it will be in the future, by the Spanish and Italian producers. Other major producing countries are Yugoslavia and Mexico.

Probably no other mineral commodity has fluctuated so much in price. A 76-lb flask of mercury sold for $195 in 1943, $76 in 1948, and $325 in 1954. This wide fluctuation has resulted in spasmodic domestic production. U. S. mines produced 51,929 flasks in 1943, and only 4535 flasks in 1950. In 1955 the estimated production was 18,400 flasks. Domestic consumption in 1955 was 57,200 flasks.

It was not until a three-year guaranteed price program was announced by the General Services Administration in July 1954 that mercury-producing companies in this country could formulate any long-range plans. This GSA program calls for the purchase of 125,000 flasks of domestic mercury and 75,000 flasks of Mexican metal. The guaranteed price for domestic production is set at $225 per flask, or $2.96 per lb. That for Mexican metal is $210 per flask, or $2.76 per lb. Termination date for the buying program is Dec. 31, 1957, unless the quota has been filled prior to that date. Since the price support program was inaugurated the price has not fallen below $255 per flask. It is reported that to date GSA has purchased only five flasks, which were sold to insure the continuation of the buying program.

Location and History: The Idaho-Almaden mercury mine is on top of a prominent topographic feature known as Nutmeg Mountain in sections 32 and 33, T. 11 N., R. 3 W. and sections 4 and 5, T. 10 N., R. 3 W., Boise Meridian. The property is about 11 air miles and 16 miles by road in an easterly direction from Weiser, Idaho, in the extreme west-central part of the state. Weiser, the county seat of Washington County, with a population of about 4000, is a stop on the main line of the Union Pacific RR near the confluence of the Snake and Weiser rivers. Agriculture is the principal industry in the area. Employes live in Weiser and vicinity and drive to and from work each day, as the mine is readily accessible by well maintained county roads.

J. R. REYNOLDS is Chief Engineer at Rare Metals Corp. of America, Salt Lake City.

Quicksilver mineralization atop Nutmeg Mountain was first recognized in 1936 by an Oregon sheepman named Harry Brown. Principal claims in the group were staked in 1937. At the present time the property consists of one patented mining claim, 32 unpatented mining claims, and one 40-acre tract of fee land, a total of approximately 600 acres. The property was examined and sampled by L. K. Requa and then leased to the Idaho-Almaden Mines Co. in 1938. This company was a closed corporation consisting of L. K. Requa, Allen Hoover, Herbert Hoover, Jr., and their wives. A 50-tpd Gould furnace and condenser system was erected in 1939 and operated continuously until 1942, when depletion of high-grade ore reserves and the low price of mercury resulted in the abandonment of the property. Approximately 4000 flasks of mercury were recovered from about 50,000 tons of ore during this operation. The property lay dormant, other than for annual assessment work, from 1942 until late in 1954 when active exploration was started by Rare Metals Corp. of America. The first examination of the property by Rare Metals personnel was made late in July 1954. Shortly thereafter negotiations were started with the present owners, heirs of the original locator. These negotiations resulted in completion of a lease and option to purchase contract on Oct. 1, 1954. Rare Metals immediately equipped a field office and laboratory and initiated exploration drilling. Between October 1954 and March 1955, 68 diamond drillholes, aggregating 1738 ft, and 451 wagon drillholes, aggregating 15,715 ft, had been drilled. More than 4000 drill samples were collected and assayed. From these data total reserves were calculated and mining and furnacing costs determined. Because the ore was relatively low grade, it was apparent that a plant capable of treating 150 to 200 tpd would have to be built to make the treatment of such ore economical.

When enough ore reserves had been developed to justify a reduction plant of this size, a number of independent contracts were let which included overburden stripping, steel tanks and ore bins, conveyors, steel buildings, plant construction, and erection. The plant was completed and the first test run made during the last week of September 1955. The time from the first examination to initial production was less than 14 months.

The Deposit: Four orebodies have thus far been developed at or near the surface of the relatively

Idaho-Almaden mercury mine—looking south from top of condenser system showing *A* orebody from which overburden has been removed. Stripping depth varied from a few feet to as much as 25 ft, 10 ft being about average.

flat-topped Nutmeg Mountain. This mountain is the most prominent topographic feature in an area characterized by a series of low, gently sloping hills covered by grass and sage. The almost sheer north and south faces of Nutmeg Mountain are probably the result of rapid erosion rather than expressions of any geological structure.

Mercury deposits at the Idaho-Almaden occur in a thick sedimentary series known as the Payette formation, which is generally considered to be Miocene in age and probably rests upon granitic rock relating to the Idaho Batholith. It is overlain by flows of Columbia River basalt. To the southwest the basalt is covered by the Idaho Formation (Pleistocene). In the vicinity of the mine, erosion has stripped off the Idaho formation and most of the basalt.

The Payette formation consists of a thick series of massive, fairly coarse-grained, arkosic-like sandstone, which is interbedded with shale and clayey sandstone. It is extremely porous with cementing material at a minimum. Basalt dikes and sills have intruded the sedimentary formations and outcrop in the mine pits and other parts of the property.

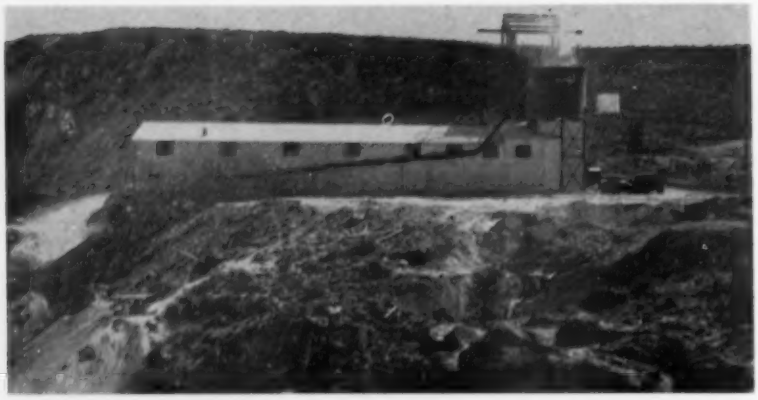
The dominant regional structure is a system of broad parallel northwest-trending anticlines with the steepest dips, often 30°, on the southwest flanks. The orebodies are associated with a well defined fault pattern that developed parallel with and also at right angles to the axis of one of these anticlines, forming a structural basin. These faults appear to be tension fractures with comparatively minor displacement. It was along these fractures that the mineralizing solution rose, creating the orebodies now being mined.

Cinnabar is the only mercury mineral recognized. It is almost always accompanied by opal or chalcedony or both. Deposition was probably from alkaline solutions and no doubt took place near the surface and at a temperature between 110° and 150°C. The solutions migrated out laterally through the more porous sandstone beds, creating large, blanket-like deposits. In addition, some of the feeder fractures were mineralized to the extent that they are commercial grade ore. Occasionally the cinnabar occurs alone as fracture fillings and cement in grain exterstices of the porous sandstone.

Development work at the Idaho-Almaden has outlined four orebodies containing a total of approximately 525,000 tons of proved ore with an average content of 3.6 lb of mercury per ton. These four orebodies are referred to as *A*, *B*, *C*, and *D*.

Drilling holes on 6-ft centers using a Gardner-Denver D-99 drill with a 380-cu ft Joeger compressor mounted on an International half-track.

Eimco 105 loading broken ore from *A* orebody. Total cost to mine and deliver to bin is approximately $1.00.



Idaho Almaden mercury reduction plant showing how hot air is delivered to ore bin for drying and preheating.

Ore ranges from 3 to 45 ft thick and was overlain by 0 to 30 ft of overburden. Overburden consisted of soil, semiconsolidated shale, and landslide debris.

Mining Methods: Mining at the Idaho-Almaden is by open pit. There are several vein-like occurrences on the property which will be mined by conventional underground methods at some future time.

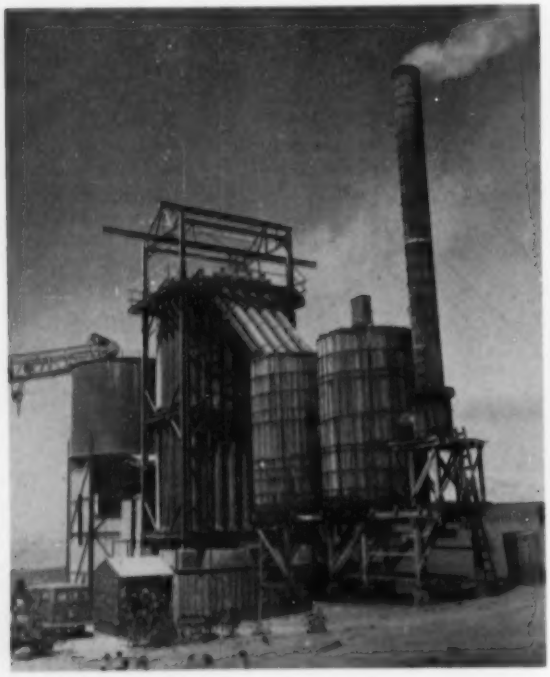
Prior to actual mining operations, the bulk of the overburden was stripped from *A*, *B*, and *D* orebodies under an independent contract at a cost of 26½¢ per yd. The rock was soft enough to require very little drilling and blasting. Bulldozers, rippers, and carryalls were used to remove approximately 100,000 cu yd of overburden. Stripping depth varied from a few feet to as much as 25 ft, with 10 ft being about the average.

Current mining practice calls for blastholes to be drilled on 6-ft centers. At this time a 24-ft bench is being carried at the *A* orebody and an 18-ft bench at *D*. Drilling is done with a Gardner-Denver D-99 wagon drill, which is mounted, together with a diesel-powered 380-cu ft Jaeger compressor, on an International half-track. Coupled steel and 2-in. tungsten carbide bits are used in blasthole drilling. Holes are loaded with Atlas-Flodyn No. 7 bag powder and fired with electric delay caps. The ore breaks easily and fragmentation is excellent. An Eimco 105 loader loads the broken ore into Chevrolet dump trucks. The ore is weighed on a Fairbanks-Morse 20-ton platform scale and delivered either to the coarse ore bin or to storage.

The total direct cost to mine and deliver one ton of ore to the coarse ore bin is approximately $1.00.

Furnacing: In general mercury is recovered from its ores in furnaces and retorts, the bulk of the production coming from furnaces. The two principal types of furnaces are the multiple hearth and rotary kiln. Under certain conditions it has been found practical to concentrate cinnabar by flotation and retort the mercury from flotation concentrate.

Ore at the Idaho-Almaden is treated in a Gould rotary kiln and condenser system. The flowsheet of an average plant is comparatively simple, and that at the Idaho-Almaden is no exception. Broken ore from the pits is trucked to either the grizzly over the coarse ore bin or to storage on a 80x80-ft concrete slab. It is the general practice to store high-grade ore from *C* and *D* orebodies on the slab and



ABOVE: Master control panel. SIDE: Condenser system consists of 76 pipes 18 ft long. The redwood tanks are 12 ft diam and 20 ft high.

to blend with the lower grade ore from *A* and *B* in order to maintain a relatively uniform feed to the furnace. A rubber-tired Michigan loader transports the ore from the various stockpiles to the grizzly. The coarse ore bin is of ⅜-in. steel construction with a sloping grizzly bottom having 3-in. openings. Material of −3 in. drops through the grizzly directly onto the conveyor to the fine ore bin. Oversize passes through a 30-in. Syntron vibrating feeder to a 20x30-in. Kue Ken crusher that is set at 3 in. Crushed ore joins the undersize from the grizzly on an 18-in. conveyor belt 186 ft long which discharges into the top of a covered 300-ton circular ore bin with a conical bottom. Ore is fed from this bin directly into the furnace by a Gould reciprocating feeder. Feed rate is governed by regulating the number of strokes made by this feeder. The feeder is so constructed that the ore forms a seal to prevent the escape of the mercury-laden gases from the furnace. The furnace itself, the world's largest single-tube mercury kiln, is 90 ft long and 5.5 ft in diam. It is set on a slope of ¾ in. per ft and rotates at 3½ rpm. The furnace is lined with sheets of asbestos 1 in. thick between the steel shell of the kiln and the 6-in. thick pinite fire brick lining. A new type of low-pressure Hawk burner supplies the heat at the lower end of the kiln. At this time No. 5 Bunker oil is being burned at the rate of 7½ gal per ton of ore. Ore passes through the kiln counter flow to the heat. Contact time in the furnace is about 45 min, during which time the ore is raised to a temperature of about 1200°F. Cinnabar decomposes quite readily when heated to about 600°F. In addition to the mercuric and sulfurous vapors, there is also considerable moisture and water of crystallization from the opalite liberated by the ignition of the ore. These gaseous products, together with varying quantities of soot and dust, are drawn from the feed end of the kiln by a 5-ft Sirroco fan and dust collector. The relatively dust-free gases pass into the condenser system, thence through two redwood tanks and into the atmosphere through a redwood stack. The condenser system consists of 76 pipes 18 in. diam and 18 ft long. The first four of these pipes are of steel and the remaining 72 of heavy-gage cast iron. The redwood tanks are 12 ft diam and 20 ft high. Stack losses appear to be negligible when the stack temperature is kept below 120°F. The bulk of the mercury condenses in the cast iron pipes.

The mercury, soot, and mud are drawn off into an automatic hoeing machine, where quicklime is added to dry up the mixture and liberate the quicksilver. The mercury then passes through a series of filters into an automatic bottling device.

Burnt rock is discharged from the main kiln into a fire-brick lined chute that extends down into a second kiln 6 ft diam and 40 ft long. This kiln serves as a conveyor to transport the hot rock out of the mill building and discharge it over the top of a nearly sheer cliff several hundred feet high. Cool air is drawn into this kiln and heated by contact with the burnt ore. Part of the heated air is drawn through a boiler where water is heated. It is planned to circulate this hot water through pipes that were laid in the concrete blending and storage slab to keep the stockpiled ore from freezing during the winter. The hot air that is not passed through the boiler is blown into the fine ore bin, where it preheats and helps dry the ore. This preheating and drying has materially increased furnace capacity and has resulted in a considerable saving in oil consumption.

Mercury bottling device and storage room. Mercury is bottled in the standard 76-lb flasks.

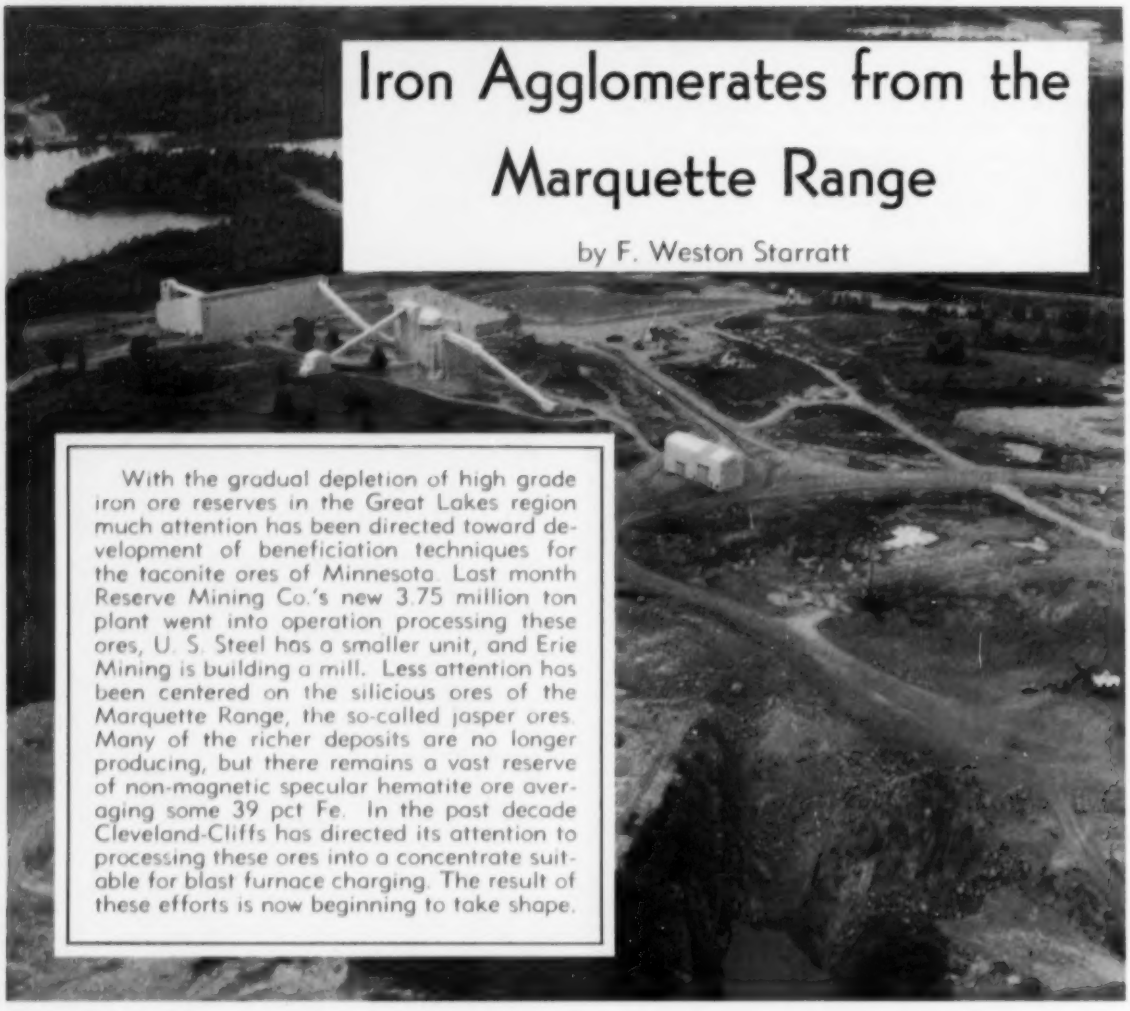
Total direct cost of furnacing averages $2.60 per ton.

Summary

Rare Metals Corp. of America has erected and placed in operation a 175-tpd Gould rotary furnace and condenser system at the Idaho-Almaden mine near Weiser, Idaho. This deposit contains approximately 525,000 tons of ore averaging 3.6 lb of mercury per ton. The four orebodies thus far developed are being mined by open pit methods. Geological possibilities of finding additional ore reserves appear to be excellent, so if the price of quicksilver remains at a level comparable to that in existence, this operation should be a long and successful one.

Iron Agglomerates from the Marquette Range

by F. Weston Starratt



With the gradual depletion of high grade iron ore reserves in the Great Lakes region much attention has been directed toward development of beneficiation techniques for the taconite ores of Minnesota. Last month Reserve Mining Co.'s new 3.75 million ton plant went into operation processing these ores, U. S. Steel has a smaller unit, and Erie Mining is building a mill. Less attention has been centered on the silicious ores of the Marquette Range, the so-called jasper ores. Many of the richer deposits are no longer producing, but there remains a vast reserve of non-magnetic specular hematite ore averaging some 39 pct Fe. In the past decade Cleveland-Cliffs has directed its attention to processing these ores into a concentrate suitable for blast furnace charging. The result of these efforts is now beginning to take shape.

THE opening of a new large-scale beneficiation plant is always news, but the start-up of Cleveland-Cliffs Iron Co.'s Eagle Mills near Ishpeming in Michigan's Upper Peninsula early in October is of particular interest because of the completely different type of operation being carried on at this plant.

The ore boat which left Marquette, Mich. on the 3rd of October bound for the plant of Wisconsin Steel Division of International Harvester, not only carried the first output of this new mill, but also carried the first agglomerate made from the Upper Peninsula's jasper ore, the first commercial output from an updraft traveling—grate furnace, and the first large-scale commercial application of the balling disk, also called the *flying saucer*, to the agglomeration of an iron flotation concentrate.

This was not the first shipment of an iron concentrate made from these jasper ores. Since the opening of the Humboldt mine and flotation mill in this same area in March 1954, the Ford Motor Co. has been taking its entire output of concentrates for Ford's sintering plant. The Eagle Mill does, however, mark the first time that the fine flotation concentrates have been made into hardened pellets for direct blast furnace charging.

Jasper ore from new Republic open pit

Iron concentrate now being received at Eagle Mills is mined and milled at Republic Mine, located on the Marquette Range some 2 miles by rail. The deposit is owned by Cleveland Cliffs, and the mill, while operated by them, is owned by a firm known as the Marquette Iron Mining Co., a venture in which Cleveland Cliffs holds 47.5 pct of the stock with the rest divided between, Jones & Laughlin Steel, International Harvester, Inland Steel, and Wheeling Steel.

At Republic, underground mining commenced about 1870, but the last shipment of direct smelting ore was made in 1937. What remained was a lower grade ore body of specular hematite, physically united with a cherty material in a stratified schistlike formation. It is covered with an overburden varying in depth up to 50 ft.

In 1947 a diamond drilling program was undertaken by Cleveland Cliffs. This proved workable ore reserves to be in excess of 100 million tons when mining by open pit techniques to a depth of 500 ft.

Republic Mill utilizes technique developed at Humboldt

The Republic mill began operations in April 1956 as the second jasper concentrating unit on the Mar-

quette Range. The first was located at Humboldt Mine, only a few miles distant. This 300,000 ton capacity operation is a joint venture of Ford and Cleveland-Cliffs. At present the output is being shipped directly to Ford, but at a later date it is anticipated that its output will also be agglomerated at Eagle Mills. The mill at Republic, with a capacity for handling some 600,000 tons of ore annually, utilizes a flow sheet similar to the Humboldt operation.

Jasper ore unsuitable for magnetic separation

Both Humboldt and Republic ores have similar characteristics, and neither lend themselves to direct magnetic separation. For some time experimental work had been conducted on a technique using a reducing roast followed by magnetic separation. This work is still being continued, but it has not proven feasible, at least for large scale production. The technique finally adopted for both plants is a flotation process similar to that frequently employed in separating non-ferrous minerals from waste. This is, so far as is known, the first time that flotation has been commercially applied to the separation of iron oxide ores.

Three step crushing operation used

At Republic, from 3500 to 4000 tpd of mill feed, averaging some 39 pct Fe, is supplied by the open pit. Mining operations use a bench height of 40 ft, with a jet piercer for primary blast hole drilling in the hard silicious ore. Broken ore is loaded by an electric shovel equipped with a 5 cu yd dipper into 34-ton diesel powered trucks for haulage to the crushing plant.

Crushing is a three step operation which reduces the ore to —½ in. The ore is discharged into a gyratory crusher, screened, and the +2 in. ore fed into a 7-ft Symons cone crusher. After a second screening, the +½ in. ore is fed into the tertiary crusher, a 7-ft shorthead cone crusher. At the rate of 500 tph the crushed ore is conveyed to the feed bin in the mill building. This bin is filled during one shift operation of the crushing unit, and it provides the mill and flotation sections with sufficient raw material for round-the-clock operation.

At Republic, two almost-identical milling-flotation units

At the head of each milling-flotation unit is a 9x12-ft rod mill which discharges into a 11-ft diam Hardinge ball mill, operating in closed circuit with a 14-ft hydroscillator. The crushed ore is ground to —65 mesh in this operation.

The hydroscillator overflow is pumped to a 24-in. cyclone followed by a bank of 6-in. cyclones for desliming and thickening. The underflow from the 24-in. cyclone discharges into a spiral densifier for further thickening. The 6-in. cyclone overflow, or slimes, which represents some 2 pct of the weight of the ore being handled at the mill, is piped to the tailing pond. While it carries some 30 pct Fe, it is much too finely ground to be treated economically by flotation or gravity methods.

The thickened product from the densifier flows by gravity to the first of four conditioners, while the underflow from the 6-in. cyclones discharges into the fourth conditioner. Flotation reagents—a fatty-acid collector, bearing the trade name Asimtol of the Arizona Chemical Co., and a frother—are added to the first conditioner in the amount of 1.2 lb per ton of ore. The conditioned pulp is pumped to the flotation cells.

Humboldt mine pit operation showing 5 cu yd electric shovel loading 34 ton Euclid truck in foreground and jet piercer producing blast holes in background.

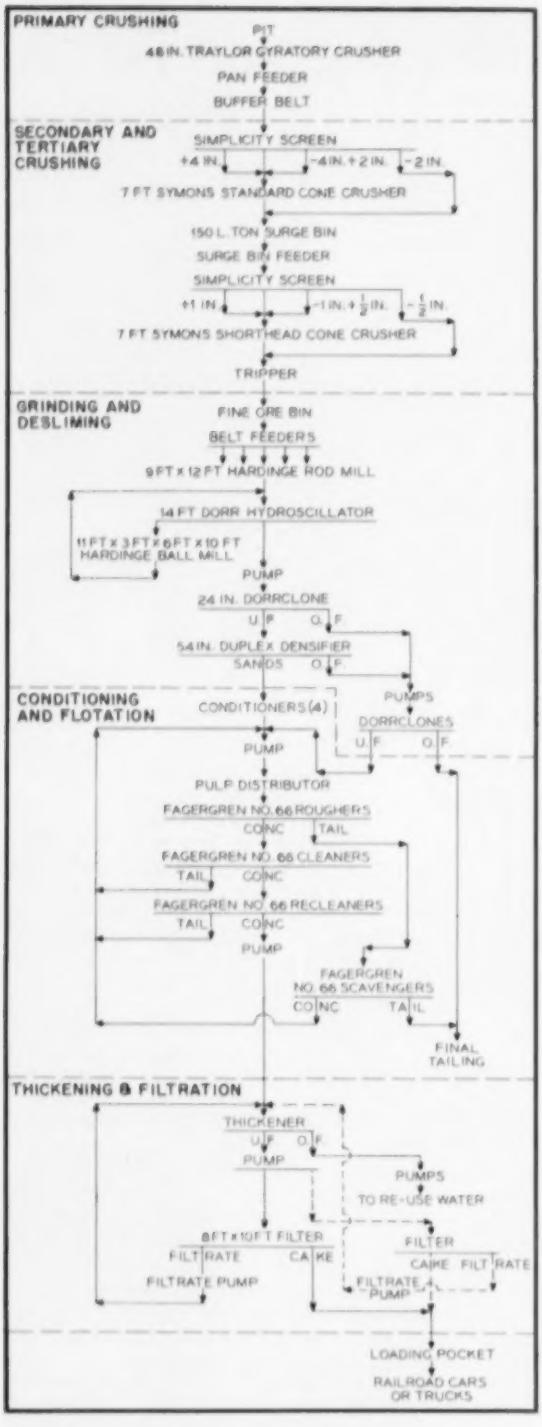
The usual arrangement of roughing, cleaning, recleaning, and scavenger cells is used. Flotation tailings carry 7.5 pct Fe, while the concentrate after passing through a thickening tank and dewatering filter, carries about 63 pct Fe, 8 pct SiO_2, and 3 pct moisture. Each flotation unit is capable of treating some 80 tons of feed per hour. In the over-all operation only some 9 pct of the available iron plus that iron chemically combined with silica is lost.

In the past few months since Republic Mill went into operation, concentrate has been stockpiled, but now, with Eagle Mills also in operation, the concentrate is discharged into railroad cars for direct transport to Eagle Mills, some 32 miles distant.

At Eagle Mills, Republic concentrate agglomerated into direct charging pellets

Eagle Mills agglomerates the —65 mesh iron concentrate into sizes sufficiently large to be handled in the blast furnace. But, to produce a satisfactory pellet it is necessary to start out with a very finely ground material—65 mesh is much to coarse! So, the first step is a grinding operation, and for this a Hardinge Tricone ball mill is used to reduce the concentrate to 65 pct —325 mesh. The finely ground material is thickened and dewatered.

The unfired green pellets are a blend of the concentrate with 2.5 pct ground limestone, 4 pct ground anthracite coal, and 0.5 pct bentonite. A Hardinge Thermomill is employed for grinding the first two, while the latter is received at the plant in sufficiently fine particles. Bentonite serves to bind together the wet particles of concentrate, while limestone promotes the formation of a low temperature slag which holds the dried pellets together in the lower temperature region of the furnace. In addition, it is presumed that the inclusion of limestone will tend to promote better blast furnace performance with these pellets.



Simplified Concentrator Flow Sheet

Marquette Iron Co. • Republic Mine • Republic, Mich.

The 500 tph crushing unit fills the fine ore bin in one 8-hr shift. Bin feeds two almost identical grinding-flotation units —one shown above—operating on three shift basis. Dewatered concentrate is transported to the Eagle Mills for agglomeration.

Balling Disks, or Flying Saucers, used for agglomeration

The concentrate, bentonite, and ground limestone join each other at the balling disks. Eagle Mills has four of these 18 ft balling disks, which revolve in a plane enclined some 50° to the horizontal. The blend is deposited by conveyor in the center, and, as it tumbles in the disk, it forms round particles which grow larger as they pass over successive lips to the outer edge of the disk. Ground coal is added in the last lip of the disk in order to coat the agglomerate with a thin layer of fuel for the heat hardening operation. The pellets discharged from the disk range in size from $+\frac{3}{8}$ in. to $-\frac{1}{2}$ in.

Pellets heat-hardened in updraft traveling-grate furnace

The design of this furnace has certain similarities to the traditional Dwight-Lloyd sintering machines, and it has, in fact, been designed by the McDowell Co. The most significant difference with this traveling grate furnace is the updraft principle which permits, not only a thicker layer of material to be treated, but also permits an extended high temperature treating of the green pellets without damaging the malleable iron grate bars. The grate conveyor is 6 ft wide and 223 ft long; the first 168 ft are the firing zone, and the remainder serves to cool the pellets. The furnace is capable of heat hardening some 2000 tpd of pellets at a maximum temperature of some 2500°F. The grate speed is 30 in. per minute.

The hearth layer for the furnace, deposited directly on the grate, consists of 4 in. of $+\frac{1}{4}$ in. $-\frac{1}{2}$ in. return pellets, screened out at the exit end of the furnace, plus a certain quantity of ground ignition coal. The layer is subjected to a maximum of 5 in. downdraft as it passes into the ignition box. Propane is used to ignite the top of the layer, and it passes on over five downdraft wind boxes which ignite thoroughly.

Further along the grate, 8 in. of green pellets are deposited on the hearth layer as it moves onto the first of 23 updraft wind boxes. The updraft causes the hearth layer to ignite the green pellets, and at later stages three more 8-in. layers of pellets are deposited. Each of these layers is successively ignited, and toward the end of the furnace, there are, thus, a total of 32 in. of hardened pellets and a 4 in. hearth layer on the grate. At this stage a draft of 20 in. is required for firing.

The heat hardened pellets are discharged onto a screening machine. Plus ½ in. goes to the waiting railroad cars, $-\frac{1}{4}$ in. is reground in the ball mill, and the fraction in between these is used for the hearth layer.

1957 expected to show full-scale pellet production

The heat hardened jasper iron pellets are transported by rail to the docks at Marquette, 7 mi distant. While only some 70,000 tons of these 63 pct Fe pellets are expected to be moved out this year, it is anticipated that next year will witness full-scale production of 600,000 tons of high grade iron pellets for the steel industry. There is a ready and waiting market for these pellets. The only direction in which this new agglomeration operation is likely to move is that of expansion!

Lightweight Aggregate - Present and Future

by Allen R. Rowen

As more is learned of its potential, manufactured lightweight aggregate is coming into widespread use. New products such as bituminous concrete aggregate are being developed. A turn-key plant is now offered, insuring future supplies and forcing down the cost of manufacturing facilities.

One of the greatest deterrents to more widespread use of manufactured lightweight aggregate is the fact that no industry-wide standards for its application exist. It is true that ASTM has specifications covering lightweight aggregate, but they are so broad as to be of little assistance in mix design work. Today even an engineer who has worked with one lightweight aggregate may be faced with re-education on his next lightweight concrete job if he is in an area served by another producer.

MANUFACTURE of lightweight aggregates has grown with the country, appearing in the patent literature at least as early as 1875, when a patent was issued covering use of burnt clay aggregate as an admixture in manufacture of artificial stone. The widely known Haydite came into the picture prior to 1920 as a result of work by Stephen J. Hayde, and there have been many others since who have contributed significantly to the growth of the industry and its technology.

Actual consumption of manufactured lightweight aggregate in large quantity by the construction industry has become manifest only since the end of World War II, particularly in the last five years.

The two processes accounting for more than 95 pct of manufactured lightweight aggregate in this country are the rotary kiln and the Dwight-Lloyd continuous grate sintering processes. Yearly output from each method is about the same in tons, although the number of rotary kiln installations is somewhat larger. In the earlier days of the industry, the relatively low output of the kilns rendered them more desirable units than the high production sintering machines. Nowadays, however, other factors are more likely to influence selection of one type of equipment over another. For instance, the sintering machine is generally considered to have a much higher thermal efficiency, and since fuel is one of the largest cost items, this is an important factor. Also it is not unusual these days for one sintering machine to produce more than 2000 tpd, with only one operator required per shift. Power consumption per unit output is usually about the same for either process and will favor the sintering method increasingly as total output expands.

The geographical distribution of manufactured lightweight aggregate plants is widespread, with virtually no area of the country entirely unrepresented. The largest plant is in North Carolina; on the other hand, the largest producer of manufactured aggregate operates plants in Texas, Louisiana, and Oklahoma.

Shovel working in open pit represents first step in production of perlite, one of the lightweight aggregates that has achieved industry stature in the past five years.

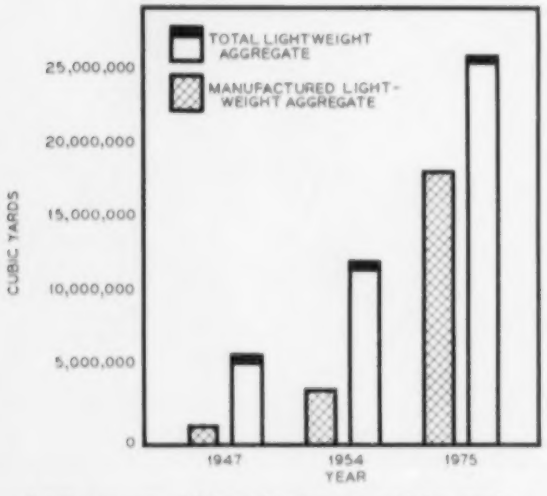
A. R. ROWEN is with the Dwight-Lloyd Div., McDowell Co. Inc., Cleveland.

Two trade associations represent the industry. The Expanded Shale Clay and Slate Institute, with headquarters in Washington, D. C., presently restricts its membership to rotary kiln producers, whereas the Expanded Shale Clay and Slate Institute of Allentown, Pa., allows membership to both kinds of producers.

At the present time the major consumer of manufactured lightweight aggregate is the concrete masonry unit producer. In fact, the immediate past and the present growth of the lightweight aggregate field has been principally brought about by requirements of the block trade for an increasing supply of aggregate in the face of otherwise dwindling sources, such as stoker ash. In 1947 the total number of lightweight blocks, in 8x8x16 equivalents, was 460,000 units. By 1954 this figure had increased to 950 million a year, and conservative estimates indicate a 2 billion unit year by 1974. Assuming an average production of 65 to 70 blocks per yard, block industry consumption of all kinds of lightweight aggregate in the above three years amounts to 6,150,000 cu yd in 1947, 12,650,000 yd in 1954, and by 1974, 26,750,000 yd.

Not all lightweight aggregate has been manufactured. Much has been byproduct, such as blast furnace slag and cinders. However, the percentage share as well as the total use both favor vast future growth. Another look at the production figures above reveals that less than 20 pct of the lightweight units produced in 1947 incorporated manufactured lightweight aggregate. By 1954 the figure was over 27 pct, with cinders amounting to 43 pct. Assuming that the cinder supply continues to decline as estimated, the market for manufactured lightweight aggregate by 1974 will be 70 pct of the total lightweight yardage, or 18,725,000 yd.

The other large volume use for manufactured lightweight aggregate is in structural concrete. Here there is far less competition from byproduct aggregates, because concrete requirements as to weight-strength ratio are often too severe for any but materials of the highest quality. It is possible to design lightweight concrete mixes with compressive strength of 3500 to 4000 psi with nearly the same cement-aggregate ratios used for heavy aggregates.



Relative amounts of lightweight aggregate needed to supply past, present, and future demand of masonry unit industry.

Very high strength designs above 9000 psi are possible, as are very low density mixes with concrete weighing less than 50 lb per cu ft. The great advantage in the use of manufactured lightweight aggregate for structural concrete is the saving in weight, although other desirable properties are often contributing factors. The low modulus of elasticity results in very tough concrete, the low thermal conductivity yields good insulating value, fire resistance is well above sand and gravel concrete, and soundness is excellent.

To return to the value of low density, there are startling examples of savings resulting from this property. The Tacoma Narrows bridge, third longest suspension bridge in the world, was reduced in floor weight by 4,120,000 lb through use of expanded shale aggregate, which brought about a reduction of 735,000 lb of structural steel and 725,000 lb of suspension cables and wire. Cost saving was $320,000. Concrete cylinders after one year were up to 9000 psi. Construction of the new Statler Hotel in Los Angeles used 50,000 yd of manufactured lightweight aggregate, reducing the dead load of the structure by 37,000 tons. This resulted in savings of 15 pct on the weight and cost of the structural steel. Cement contents per cubic yard of concrete were from 5¾ to 6¼ sacks, depending on the application, and compressive strengths ranged from 3500 to 4800 psi.

The industry as a whole is combatting this situation in two ways. Its trade associations are sponsoring extensive work at several universities and research institutes to establish standards of application as well as of manufacture. Although these studies are long range, they have already produced valuable data soon to be published, and they will in time establish specifications that will allow the use of lightweight aggregate with no more difficulty than that presented by conventional sand and gravel mixes, and in some cases, with fewer problems.

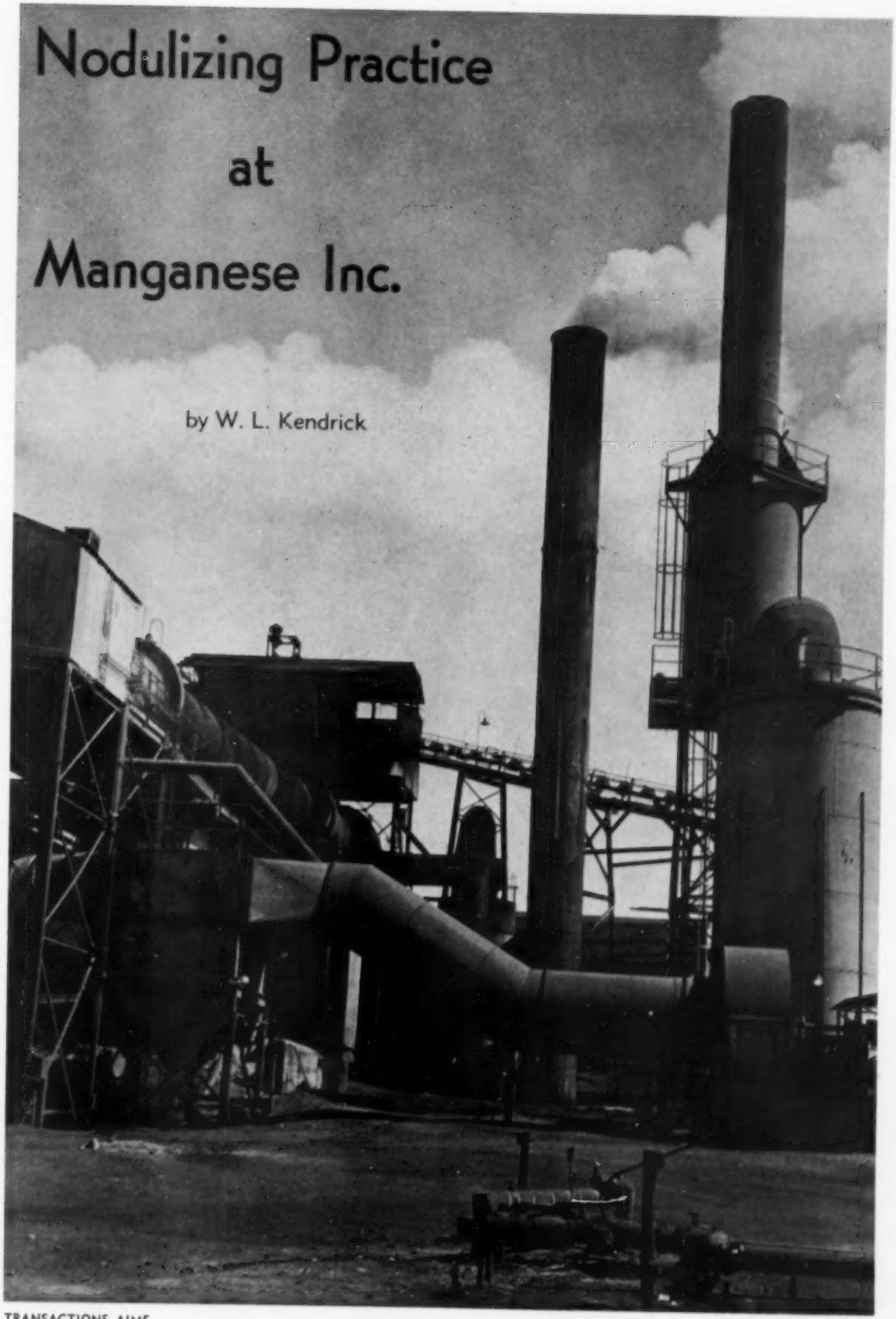
The second and more immediate step being taken by the producers individually to simplify use of their product is to make available a competent application engineer to recommend and assist in mix design and concrete replacement.

Although at present most lightweight aggregate is used in conjunction with Portland cement, another market is under development which will be equally large, if not larger—bituminous road concrete. Studies show that a highly skid-resistant surface can be obtained by including lightweight aggregate, properly graded, in the bituminous mix. Another very important property imparted by the aggregate prevents the bituminous road surface from creeping as it is laid on or even after long heavy usage. Should this field continue to develop as indicated, the number of planned and existing lightweight aggregate manufacturing facilities would be far from adequate to meet the demand.

One of the principal reasons there are not more lightweight aggregate plants is the high cost of building and placing in operation an entire facility. Most existing plants have been built with little or no outside help, and perhaps not as economically as they might have been. To counter this, and to bring the entire cost of a plant into focus before the project is undertaken, a new concept in this field is being offered. A complete plant, engineered, constructed, and ready for operation can now be bought for one contract price, so that no unknown factors will spring up to cripple the operation before it starts.

Nodulizing Practice at Manganese Inc.

by W. L. Kendrick



The Transactions papers appearing in Mining Engineering *during 1956 will subsequently appear in AIME Transactions Volume 205, and may be permanently referenced to that volume.*

AT Henderson, Nev., Manganese Inc. is mining by open pit methods a low grade manganese ore averaging 21 pct Mn, concentrating it to 42 pct by flotation, and agglomerating it to produce nodules of metallurgical grade—about 48 pct Mn.

The striking aspect of the whole operation, and one that has considerable effect on nodulizing, is the use of approximately 250 lb of reagent per ton of mill feed.

For many years rotary kilns have been used for drying, calcining, clinkering cement and other products, and burning lime. Their use for nodulizing or agglomerating a metallic ore is new—first reported in this country in 1904, when a small rotary kiln at South Chicago was employed to nodulize flue dust from a blast furnace. After a short time this process was replaced by sintering.

During World War II the agglomerating process gained considerable impetus, chiefly because there was a shortage of open hearth ore for the steel industry, and several operations were set up to nodulize fine iron ores in kilns that were available, usually cement kilns. Since the war several installations have been set up for nodulizing or agglomerating various products. One of these is the plant at Henderson, Nev.

The process of nodulizing, in a rotary kiln, is simply heating material, generally finely ground, to a temperature at which the product becomes plastic. By the rotary action of the kiln it then rolls into balls and sticks together to form larger balls. The ultimate aim of the operation is, of course, to make balls of the desired size without having too much of the material stick to the walls of the kiln.

There are three important factors for satisfactory kiln operation:

1) Constant feed of a given tonnage, each minute of each hour of each day.

2) Constant chemical analyses of the feed so that the proportion of each element or mineral, including moisture, is always the same.

3) Constant temperature and draft so that the same condition always exists in the firing zone of the kiln.

When the first two conditions are achieved, the third is possible.

At Manganese Inc. none of these conditions exists. There is also the problem of eliminating lead from the feed to meet certain specifications in the nodules.

Mill concentrates to be fed to the kilns vary from 39 to 44 pct Mn, 7.5 to 12 pct silica, and 1.0 to 4 pct Pb, with varying amounts of aluminia and alkaline earths. Because of the extremely large amount of reagents used, the most economical way to get concentrates to the kilns has been direct filtering, in leaf filters, in circuit with a thickener and cyclones. These filters produce a very wet sticky cake that averages 18 to 21 pct reagents, consisting of diesel fuel, soap and oronite, and 20 to 25 pct moisture. It is difficult, to say the least, to produce a steady, uniform feed for the kiln directly from the mill, since feed varies not only with mill operation but also with variations in ore fed to the mill. Operation of the kilns must be regulated from hour to hour and day to day to meet the changing nature of the feed.

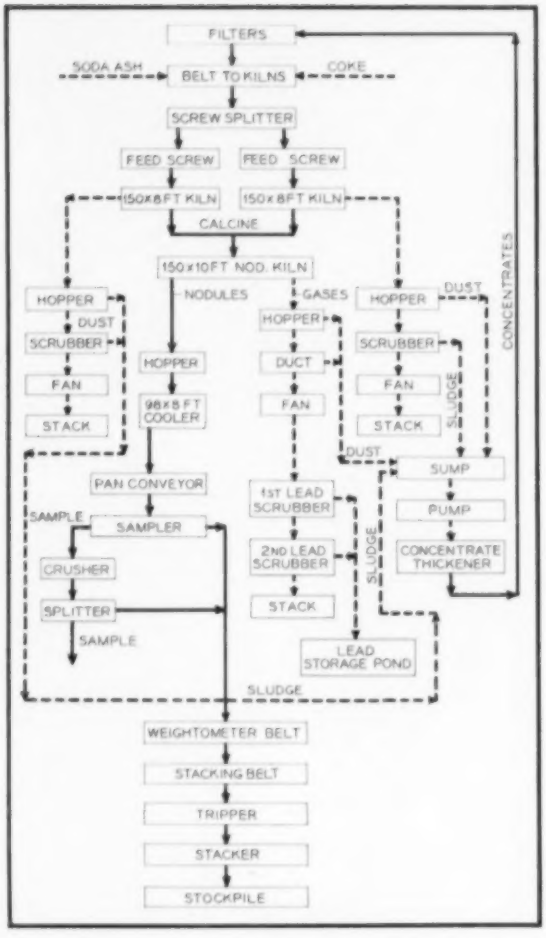


Fig. 1—Flowsheet, nodulizing section, Manganese Inc.

On the flowsheet, Fig. 1, progress of the material can be followed from the filter building, through the nodulizing section, to the final stockpile.

At the filter building petroleum coke is added at approximately 3 pct of the dry weight of cake, and soda ash at 0.3 to 1.0 pct. The coke is added as a reducing agent in the nodulizing kiln, and the soda ash helps pelletize the material and also cuts down on dust.

This combined feed is elevated by conveyor belt to the feed end of the first two kilns, which are in parallel, where it is split by a double screw feeder. Each screw can be varied in speed, and baffles in the chute above can be shifted to divert the feed to one or both screws so that it can be regulated to each kiln. From this splitter the feed drops to a ribbon screw (18 in. diam and 18 ft long) in the end of each kiln which carries the feed through the dust hood and deposits it in the kiln 5 ft from the end. Eight chains, 3 ft 6 in. long, hanging with a loop of about 10 in. in the middle, are staggered over 5 ft of kiln length at the feed point to prevent the sticking of the greasy, wet material and the formation of a mud ring.

Within the first 30 or 40 ft of these kilns most of

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the reagents burn off at a temperature of 800° to 1000°F. Capacity of the whole operation is governed by the fact that reagents can burn off at this point—which in turn depends on the amount of oxygen that can reach this point. At the discharge end the two kilns are fired with preheated bunker C fuel oil by boiler-type panco burners, which are used to heat up the kilns and, under normal operation, to keep the fire going in the feed end. There is usually enough fuel in the feed to provide sufficient heat to bring the material up to the discharge temperature of 1000° to 13000°F. Fuel consumption is about 2 gal per ton of dry feed.

The draft for each of these kilns is produced by a 48,000-cfm fan. This draws the gases from the feed end of the kiln through a dust hood and duct to cyclonic scrubbers (10 ft diam and 24 ft high) with a duct from the top to the fan and thence to a stack 100 ft high. A very incomplete combustion takes place and a dense black smoke is produced. The scrubbers as well as the duct and fan contain sprays to cool the gases and remove most of the dust. The sludge produced in the scrubbers as well as the dust from the hoods on the feed end of the kilns is collected in a sump and is pumped back to the thickener at the filter building to be returned as a circulating load.

A very greasy sludge, locally called *goop*, collects in the bottom of the fan housing and in the stack. This must be removed about every half hour and hauled to a stockpile, where it eventually dries out to be returned to the thickener. This goop represents the chief source of loss in manganese, as it contains about 10 pct Mn.

The discharge of these two kilns is usually in the form of small pellets, with a particle of coke in the center and the fine ore held to it by the sticky soda ash. There is decided reduction in oxygen content of the manganese oxides during this operation, and 50 to 65 pct of the discharge is in the form of MnO.

This material is fed to the nodulizing kiln through two stainless steel pipes, which deposit it 3 ft from the feed end of the kiln. The nodulizing kiln, 150 ft long and 10 ft diam, has an enlarged or belly section 11 ft 6 in. diam and 20 ft long (with transition sections 8 ft long) that is 28 ft from the discharge end.

In this kiln the material is heated to the nodulizing temperature, which varies from 1800° to 2150°F, depending on the silicia content, and/or the ratio between the silicia and the alkaline earths. At about 1600°F, or near the middle of the kiln, the coke becomes active as a reducing agent and the lead fuming starts. This reaction continues to within 3 to 5 ft of the discharge end. The lead in most of the Henderson ores is in the form of a lead manganate—coronadite—and just what reaction takes place for liberation is not definitely known. One of two reactions is assumed, either of which, or both, may be right or wrong. The lead may be reduced to metallic lead by the reducing action of the coke and fumed off as an oxide or a sulfate by combining with the oxygen or sulfur dioxide in the gases or it may simply be replaced in the coronadite by one of the alkaline earths, which are generally present as sulfates, and be fumed off as a sulfate. It is definitely known that both lead oxide and lead sulfate are present in the lead sludge knocked down from the exit gases of the kiln and also that there must be a reducing reaction and then an oxidizing reaction in the kiln to produce good lead elimination. This latter condition has been found necessary both in kiln operation and in laboratory work. The lead in the ores at Manganese Inc. is so intimately and uniformly integrated with the manganese that it is impossible to segregate a sample of the lead mineral for study or test. The presence of the mineral coronadite has been confirmed only by X-ray analysis and no other lead mineral has been found.

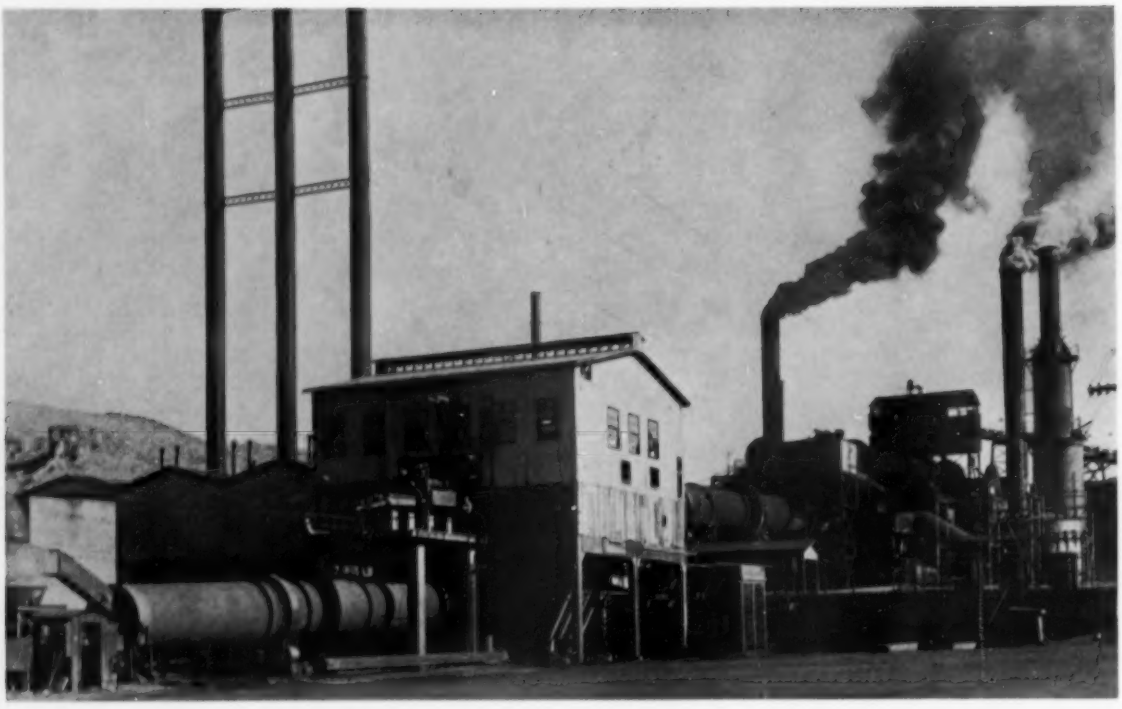
Primary and secondary lead scrubbers.

Practice has shown that there must be 2.75 to 3.25 pct coke in the feed to obtain satisfactory lead elimination. Any deviation from this range results in poor lead elimination and a decrease in upgrade of the manganese.

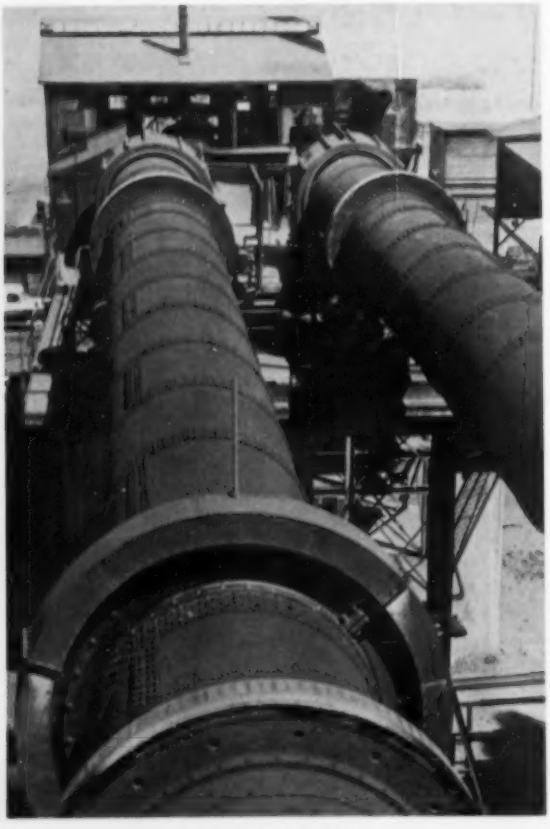
Within the range of the fuming temperature, a very strong exothermic reaction also takes place, during which the MnO produced in the first kilns changes back to MnO_2 or Mn_2O_3 or possibly other oxides of manganese, as manganese is multivalent (valances of 1 to 7). This reaction may well enter into the lead release also.

The combination of this exothermic reaction and the burning of the coke, together with the heat of the burner, rapidly brings the temperature of the charge up to the plastic state. Thus firing the kiln and controlling the position of this plastic zone is extremely difficult, even under the best conditions. With changes in rate of feed and variations in the silica content, resulting in a change in nodulizing temperature, it becomes virtually impossible to hold the plastic zone within reach of the boring bar.

In all nodulizing operations, as well as in many



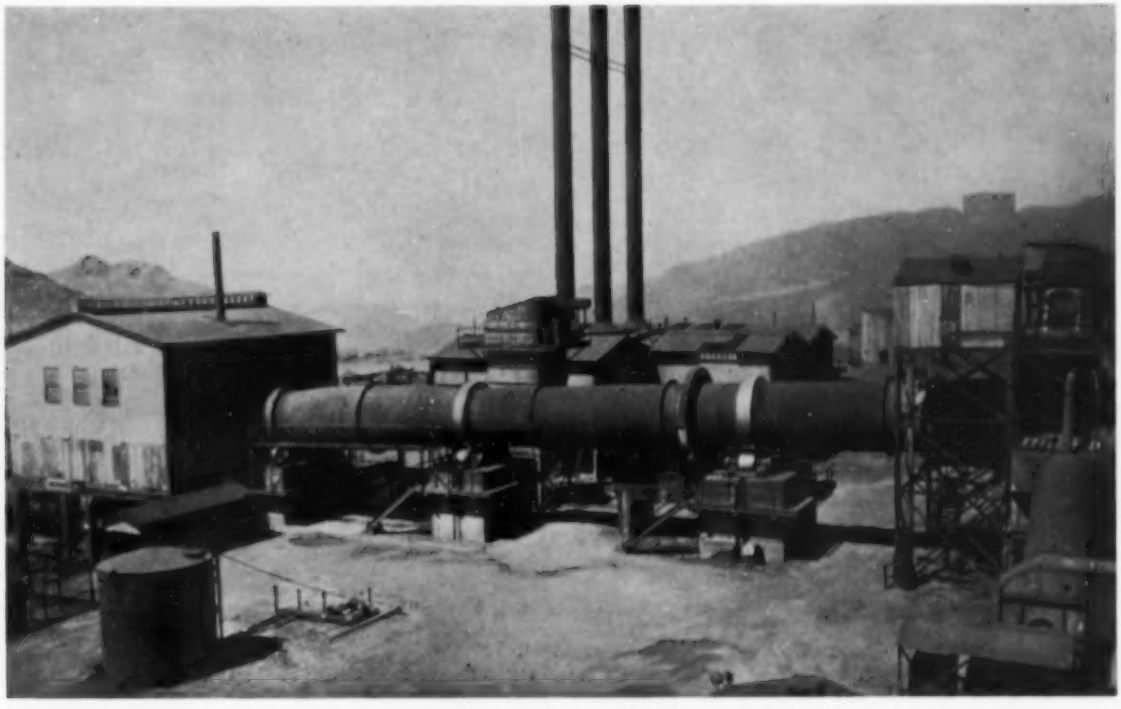
Kiln plant. From left to right: cooler, nodulizing kiln, and two calcine kilns. Lead scrubbers at extreme right.



Calcine kilns, 8x150 ft.

sintering or clinkering operations in rotary kilns, some of the material, as it reaches the plastic state, sticks to the kiln and forms rings. Manganese Inc. has this condition. Twice during a shift a boring bar is used which extends into the kiln 22 ft. Under good conditions most of the build-up is removed, but rings do form beyond the reach of the bar. Feed is shut off about once a week for an hour so that as much of the ring as possible can be shot out. Effective shooting cannot be done with feed in the kiln, as lead fuming makes it impossible to see into the kiln more than 10 or 15 ft. Every 45 to 75 days it is necessary to shut down the operation, cool the kiln, and go in and remove the coating by blasting. As the build-up becomes progressively thicker, control of the kiln is more difficult and there is a marked decrease in effectiveness of lead elimination.

To return to the flowsheet, the gases from the nodulizing kiln, containing the lead fumes, are removed by a 58,000-cfm fan through a dust hood and through an enlarged duct, which was originally a multicone, and are forced through two lead scrubbers, one above the other, where a multitude of fine high pressure sprays knock down the dust and most of the lead fumes. Resulting sludge from these scrubbers contains 30 to 45 pct Pb and 15 to 20 pct Mn. This is pumped to a settling tank, where it is allowed to dry and is then shipped to the smelter. The dust from the feed end hood as well as that from the duct is collected in hoppers, conveyed to a sump by screws, and sluiced to the same sump and pump that handles the dust from the first two kilns. It is then returned to the thickener. The total dust collected from all sources returned to the thickener and to the circuit represents 10 to 11 pct of the dry feed.



Nodulizing kiln, 10 ft diam x 150 ft long. Note enlarged section at left.

The nodulizing kiln is fired by Coen burners, using preheated bunker C oil at 200 to 400 psi. A center burner is normally used, but at times there is also a wing burner with a small intense flame directly above the charge as it rolls in the kiln. Fuel consumption is 6 to 6.5 gal per ton of dry feed. Firing temperature is indicated and recorded by a Ray-O-Tube. As previously mentioned, the nodulizing temperature varies with the silica content of the feed. The kiln is fired by eye, so to speak, and the temperature kept at the point where the charge will nodulize properly and not become too sticky and build up in the discharge hopper. As no previous analysis of the feed comes direct from the mill, no predetermined temperature can be used.

The draft of the kiln is controlled by a damper on the fan and by bleeders in the duct between the dust hood and the fan. Usually the kiln is operated at almost zero draft, just enough to remove the fumes and products of combustion. Likewise, the hood on the firing end is under no vacuum and little air enters the hood or kiln except that supplied by the primary air to the burner. A reverse draft on the cooler helps keep this firing hood at zero draft. A fume hood over the seal between the kiln and the hood has a 35-ft stack. During normal operation a small plume of fumes is evident from this stack.

The nodules produced in the kiln drop into a hopper, where a small amount of water is sprayed on them, and then pass into a rotary cooler 8 ft diam and 97 ft long. Here they are cooled to 400° to 500°F and are discharged to a pan conveyor, where additional sprays cool them to 200°F while elevating them to the sample house. A sample is cut automatically from the flow every 5 min and crushed and split. The reject is returned to the flow, which drops onto a short belt and over a weightometer before passing to the stockpiling system. A long conveyor with a movable tripper and stacker deposits the final product in stockpiles. A grizzly at the discharge of the cooler removes all the +6-in. material, which is broken up by hand and returned to the grizzly. A separate pile is made for each 24-hr period. These piles are blended as they are loaded into trucks, which haul the product to the railroad seven miles away.

Capacity of the nodulizing section at Manganese Inc. is practically 40 short tons of wet feed per hour. This represents about 22 tons of dry mill concentrates, from which 15 short tons of nodules are produced per hour. The nodules average 48.00 pct Mn, 11.50 pct SiO_2, 2.40 pct Al_2O_3, 0.60 pct Pb, and less than 0.20 pct Zn and Cu combined.

Performance of the plant can best be illustrated by data in Table I, which covers the first three months of 1956.

Table I. Plant Production at Manganese Inc., January through March, 1956

Delays	Operating Time, Hr		Percent	
	1,992.0		91.4	
Cooling and heating	69.5		3.2	
Removal of buildup	88.5		3.8	
Brick work	26.5		1.2	
Electrical	1.5		0.1	
Mechanical	4.0		0.2	
Miscellaneous	2.0		0.1	
Total delays	192.0		8.6	
Total time	2,184.0		100.0	
Production and Performance	**Short Tons**	**Long Tons**	**Mn, Pct**	**L.T. Units**
Wet feed	77,237.6			
Dry feed (concentrate)	36,181.4	32,304.9	42.25	1,364,870
Cal. dust to storage	375.1	334.9	40.00	13,396
Net feed to kilns	35,806.3	31,970.0	42.27	1,351,474
Nodules produced	29,680.5	25,500.5	48.35	1,281,242
Wet feed per hr	38.7			
Nodules per hr	14.9			
Upgrade of Mn, pct	14.4			
Feed to nodules, ratio		1.207:1.000		
Recovery, wt pct		82.9		
Manganese recovery, pct		94.8		

Discussion of this paper sent (2 copies) to AIME before Jan. 31, 1957 will be published in MINING ENGINEERING and in AIME *Transactions*, Vol. 208.

Oxidation and Enrichment of the Manganese Deposits of Butte, Mont.

by Paul L. Allsman

Butte mining district contains extensive manganese vein deposits forming a peripheral zone. Oxidation in the veins studied usually extends to a depth of about 75 ft. Secondary minerals formed by oxidation were found to be ramsdellite—always accompanied by intermixed pyrolusite—and cryptomelane. Enrichment of the gossan is accomplished by reduction of weight upon oxidation; theoretical enrichment is 32.2 pct. Additional enrichment is caused by leaching of soluble minerals, particularly calcium and magnesium carbonates.

BUTTE mining district contains extensive manganese vein deposits in the outer zone, surrounding the copper and zinc deposits and corresponding to the well known silver zone. This article describes the mineralogy of the manganese veins, the oxidation and enrichment processes, and the use of this information in prospecting.

Information was derived from a study of the Emma, Star West, Tzarena, and Norwich mines, selected as representative of the district. Vein exposures at these mines were mapped, studied, and sampled on the outcrops and throughout the oxidized zone. Specimens were cut and polished for mineragraphic examination, identification, and textural studies.

Knowledge of the manganese oxide minerals is scanty, previous information having been rendered obsolete by publication of the first correctly identified list of manganese oxide minerals by Fleischer and Richmond in 1943. Positive identification of the manganese oxides is possible only by X-ray analysis. Identifications for this study were made by the author with a Phillip's Diffractometer at the Montana School of Mines and confirmed by Lester Zeihen of The Anaconda Co., using a Norelco X-ray camera. It was necessary to re-evaluate some X-ray data, as published patterns of several manganese oxides proved to be of mixtures, mostly showing pyrolusite as a contaminant.

Perhaps the most useful information on oxidation and enrichment of manganese is presented in recent books by Goldschmidt[1] and Rankama and Sahama.[2] While their hypotheses are not conclusively proved, all laboratory and field evidence has served to substantiate them. This information was very useful in this study.

Mineralogy: The primary minerals of the manganese veins are chiefly rhodochrosite and quartz. Rhodonite is abundant in the northern part of the district and in places has been found to comprise over a third of the vein matter. A variable but generally small amount of sulfides may be present, principally pyrite and silver minerals. Sphalerite is progressively more abundant near the zinc zone.

Rhodochrosite is believed to form complete isomorphous series with siderite, ankerite, and calcite. Some variation into these compositions is common, and the intermediate forms are termed manganosiderite, manganankerite, and manganocalcite. Much of the rhodochrosite is remarkably pure. Other manganese minerals in the district include huebnerite, alabandite, and helvite.

Ramsdellite (MnO_2, orthorhombic) is the principal manganese oxide mineral, comprising perhaps two-thirds of the total oxides. It is dull to iron black, and generally massive or platy in structure. A prominent platy cleavage is the only distinguishing megascopic characteristic.

Pyrolusite (MnO_2, tetragonal) is next most abundant to ramsdellite, with which it is usually intimately mixed. The luster is often brighter or more metallic than in ramsdellite, and needle-like crystals are diagnostic. Pyrolusite is common in small cavities formed by oxidation of pyrite grains. It is relatively abundant in zones of high limonite content.

Cryptomelane (KMn_8O_{16}, tetragonal ?) is rare in the outcrop, but becomes more abundant with depth. At depths of several hundred feet it is the principal oxide. Although its appearance varies, a blue-black flinty luster and blocky to conchoidal fracture are most common. A potassium flame test will identify this mineral.

Hardness of all three oxides varies from 2 to 6. The three are quite commonly intermixed, and their textures can vary greatly. The commonest textures are massive or colloform, representative of colloidal deposition, or vuggy and boxwork textures, formed by partial leaching and oxidation in place. A boxwork of either ramsdellite or chalcedony is formed after rhodochrosite rhombs and is indicative of ore shoots in this district. Some replacement of both quartz and the granitic wallrock by ramsdellite has been noted, but most of the oxide was deposited as a fissure filling by fine particles.

No trace of manganite, hausmannite, braunite, or manganosite was found. No minerals of the psilomelane group were detected besides cryptomelane. Amorphous MnO_2 was found at several spots. A specimen of oxide coated with yellow barite crystals was amorphous and not psilomelane ($BaMn_9O_{18}$. $2H_2O$). Voids formed by the leaching of sphalerite were coated with cryptomelane, not hetaerolite ($ZnMn_2O_4$) as might be expected. No manganese sulfate minerals were found in the gossans; however manganese alum (apjohnite ?) has been re-

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TP 4359I. Manuscript, Feb. 13, 1956.

ported from the 3400 level of the Mt. Con mine, undoubtedly a secondary deposit from strongly acid copper-water.

Laboratory Experiments: Laboratory experiments included mineragraphic examination and etch tests on polished sections and tentative megascopic identification of 200 specimens. Analyses of mine water were made to determine the impurities, and particularly the pH. The Butte ground water was found to have a constant pH of 7.5, excepting the acid copper water.

The basic hypothesis of Goldschmidt (Ref. 1, pp. 621–624) regarding the weathering cycle of manganese is that divalent manganese is leached by carbon dioxide-bearing water mainly as the bicarbonate

$$MnCO_3 + CO_2 + H_2O \rightleftharpoons Mn(HCO_3)_2.$$

To substantiate this hypothesis powdered rhodochrosite was dissolved in water of pH 7.5. Mn^{++} and HCO_3^- ions were produced, as well as some MnO_2. Mine water analyses showed both Mn^{++} and HCO_3^- ions present, but no $CO_3^=$ was detected. Although this is not definite proof of the above reaction, it is sufficient to establish the reaction as plausible.

In another experiment fresh rhodochrosite faces were exposed to the elements for six months. No trace of oxidation was produced, showing that oxidation is not as rapid as often believed.

Oxidation: Assuming that the hypothesis of Goldschmidt is valid, the first step in oxidation is leaching of divalent manganese as the bicarbonate

$$MnCO_3 + CO_2 + H_2O \rightleftharpoons Mn(HCO_3)_2.$$

The second step, also postulated by Goldschmidt, is precipitation of trivalent and quadrivalent manganese as oxides. This step is dependent on both the availability of oxygen and the pH of the solutions. A pH of 7 to 8 is stated by Goldschmidt to provide the very highest oxidizing potential for manganese. The pH of 7.5 found in Butte is thus an optimum point and explains why only completely oxidized (quadrivalent) oxides are known. The reaction is

$$Mn(HCO_3)_2 + O \rightarrow MnO_2 + H_2O + 2CO_2.$$

Precipitation of MnO_2 is very nearly stoichiometric, the MnO_2 being finely divided particles, almost colloidal in size. The MnO_2 of this reaction is believed to be the mineral ramsdellite, whose textures give every evidence of colloidal deposition. Ramsdellite will invert to pyrolusite above 300°C, and is probably an unstable form of MnO_2. The intimate mixtures of these two minerals commonly found are believed to represent partial recrystallization of ramsdellite to pyrolusite.

An oxidation process of minor importance is thought to be due to the action of sulfuric acid, formed during the oxidation of pyrite. Because of the common association of zones of high limonite and high pyrolusite content, this reaction is thought to produce pyrolusite rather than ramsdellite. An intermediate step may be the formation of $MnSO_4$. However, if this reaction occurs the effect is very localized, for sulfuric acid is rapidly neutralized by rhodochrosite. At any rate the small amount of pyrite is insufficient to produce any major amount of oxidation; in fact fresh pyrite is often found as remnants in completely oxidized MnO_2.

The mineral cryptomelane is believed to be formed by absorption of potassium into MnO_2 gels, produced originally by colloidal deposition. This explains its abundance at depth, where potassium is more plentiful in circulating water than at the outcrop. Apparently potassium is absorbed more readily than either barium or zinc in Butte.

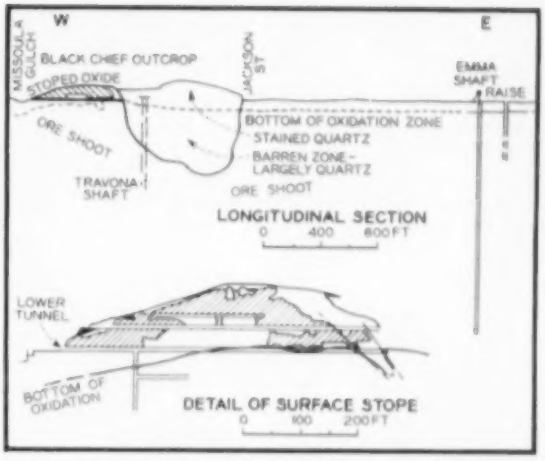

Fig. 1—Detail of surface stope (after Gidel and Steele) and generalized longitudinal section of the Emma vein.

Formation of MnO_2 from the bicarbonate requires oxygen. This is abundantly available above the ground water level, which is about 75 ft deep in Butte. The manganese veins are always more or less completely oxidized to this depth but sometimes contain a small amount of remnant protected carbonate clear to the outcrop. Occasional oxide zones extend several hundred feet below the water table, localized along fault zones which have channeled oxygen-bearing surface water to greater depths.

Manganese oxide is often formed in new workings in a few days or weeks. It is caused by $Mn(HCO_3)_2$ in solution suddenly gaining access to oxygen and precipitating MnO_2 on the walls. This is a thin skin and serves to protect underlying rhodochrosite from oxidation.

Enrichment: Oxidation of $MnCO_3$ to MnO_2 produces a reduction of weight, with a corresponding increase in grade of 32.2 pct. In general the gossan is that much richer in manganese than the primary ore. A small amount of $Mn(HCO_3)_2$ or MnO_2 is washed into the wall rocks or lost in other ways. However, such disseminations are estimated to constitute less than 1 pct of the total, the rest of the manganese being immobilized as quadrivalent oxides very near the point of solution.

While the manganese and quartz content of the vein remains relatively constant upon oxidation, most other minerals are subject to leaching and transportation by ground water. Chief among these are the calcium, magnesium, and iron content of the rhodochrosite, or sphalerite and pyrite. Leaching of such constituents produces local enrichment in excess of the 32.2 pct formed by oxidation, but the total amount of enrichment produced in this way is not large.

A phenomenon noted in the Norwich mine is rhodochrosite disseminations near veins in the primary zone. These are termed *manganese paint* and have the appearance of chalky slimes coating fractures and joints in the wallrock, growing stronger nearer the vein. This is caused by solution and reprecipitation of secondary $MnCO_3$ in the absence of oxygen, below the ground water level. It may cause some locally enriched pods, especially near faults which serve as major watercourses.

The three foregoing processes are the only enrichment phenomena of any economic significance in these deposits. In a few places small enriched zones are selectively mined, and the increased grade is always welcomed by the mine operator. But in general secondary enrichment of manganese is of little practical importance in this district.

Small deposits of MnO_2 are common along water courses, such as fissures or clay seams, several hundred feet deeper than any supergene oxides. These are due to deposition of colloidal suspensions of MnO_2 from water circulating down from the oxide zone. Such deposits give a direct indication of the total amount of secondary enrichment produced in these veins by descending meteoric water, and it is almost negligible. A typical deposit contains 4 to 6 pct manganese and occurs as scattered pods less than 6 in. thick. They are not known to form commercial deposits.

The Emma Mine: The Emma mine is the premier manganese mine in the U. S., having long supplied from 60 to 97 pct of the domestic manganese production. The Emma vein is about 40 ft wide and mining has extended to the 1700 level without bottoming the ore. The first manganese mining in Butte was carried out in 1917 by the Clark interests, on the Black Chief outcrop of the Emma vein. A detail of this stope is shown in Fig. 1. Current mining is at the rate of 1000 tpd, and total production has been 6,600,000 tons of manganese carbonate ore. The mine is operated under contract by The Anaconda Co.

The relation of the gossan to the primary ore is strikingly shown in the Emma vein, as diagrammed in Fig. 1. An ore shoot is indicated in the outcrop by high grade oxide and particularly by abundant boxworks. An adjacent barren part of the vein is obvious in the outcrop from black-stained quartz with no boxworks. This resistant quartz rib is largely responsible for the 140-ft hill of the Black Chief outcrop.

In a raise near the Emma shaft oxidation was complete to 50 ft depth. Below this no oxidation was found. The change may be abrupt, sometimes occurring within ¼ in.; however there is usually a zone of intermixed oxides and rhodochrosite.

The bottom of oxidation follows the groundwater level in a general way, but local irregularities, controlled by the local structures, are very common. In the section in Fig. 1 it is seen that descending water has carried the oxidation deeper between two faults.

Prospecting: It is believed that the chief value of this study lies in developing a technique for prospecting. Some of the Butte district is overlain by rhyolite flows, and some is covered by alluvium. However, study of the vein outcrops contained in a large area of the manganese zone would be a great aid in the development of the deposits.

Dissolving of carbonate and precipitation of oxide is believed to occur so rapidly that transportation of manganese is nil, generally producing oxidation in place. The extreme immobility of quadrivalent manganese makes any further change after oxidation unlikely. The amount of oxide removed from the zone of oxidation is easily noted in the field and is estimated to be less than 1 pct. Consequently the manganese content of the gossan is nearly the same as the manganese content of the primary vein that produced it. The primary vein can thus be accurately reconstructed at the surface.

To convert a gossan grade to the corresponding primary grade the following method is suggested. Add 1 pct to make up for dissemination of manganese into the wall rock. Then divide by a factor of 1.322. This will give a reasonable estimate of the manganese content of the primary vein at that point.

The gossan is frequently enriched by leaching of soluble constituents, as well as by oxidation, and a correction must be made. The only way to estimate the amount of leaching that has occurred is to obtain the weight of a piece of pure manganese oxide having the texture indicating oxidation in place—a spongy or boxwork texture. It is suggested that a cube of this gossan be sawed out and weighed. One cubic inch of MnO_2 formed from pure $MnCO_3$ should weigh 43.3 g. Any reduction in weight below this value is proportional to the amount removed by leaching, and the gossan grade should be lessened accordingly before changing it to primary grade. If no other evidence is at hand the leached constituents are assumed to be the calcium and magnesium content of the rhodochrosite.

Limonite indicates pyrite if it occurs as vug fillings, and siderite or ankerite if it is uniformly disseminated. Quartz is stable and relatively insoluble. It indicates a quartz-rich primary vein, which frequently forms a prominent but usually barren outcrop. Silver often leaves traces, such as the characteristic greenish to purple stain.

Careful analysis of the textures and mineralogy of the gossan will enable a prediction of the primary mineral assemblage of the vein. The grade can then be calculated, and a complete reconstruction of the primary veins can be made at this most accessible of all levels. This knowledge of ore grade and character, coupled with structural mapping, knowledge of the district habit of ore shoots, and any other geologic information available, should make possible a fairly accurate prediction of the manganese ore content of a vein in depth, in the Butte district, before any underground development is undertaken.

Acknowledgments

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Predicting Size Distribution in Classifier Products

I. MOST classifiers in use today are, in function, settling pools. A fluid suspension of particles is passed through a pool at such a rate that only a fraction of the particles—the coarser fraction—has time to settle out of suspension. The rest are carried on out of the pool as overflow. The most fundamental way in which such classifiers differ is in the mechanism used to remove the settled particles from the pool. The particles may gravitate from the pool, as in the various *sand cones*, or they may be conveyed mechanically from the pool by drags, spirals, or reciprocating rakes. The mechanism of settling pool classification and its limitations are discussed.

II. A METHOD for calculating countercurrent classifications is presented. The practical validity of the calculation method has been confirmed through 20 years of use within The Dorr Co. During this time it has been used on diverse feeds and has been checked against many types of settling pool classifiers, with close agreement in nearly every case. Excellent elimination of third critical mesh particles and finer ones may be obtained in a countercurrent classifications series where all stages separate at the same mesh, but elimination of first and second criticals is inherently limited. Spectacular elimination of all criticals may be obtained by arranging successive stages to classify at successively coarser mesh. Improvement in elimination is obtained at the cost of an increased internal recirculation, but this may be tolerable in many cases.

by E. J. Roberts and E. B. Fitch

Part I. The Mechanism of Settling Pool Classification and Its Limitations

THE mechanism of classification by settling pools is most simply shown in the case of batch sedimentation such as was analyzed by Oden.[1] The batch model will be considered, therefore, and it will be shown that the same mechanism applies to continuous operation.

Classification in Batch Settling Pools: Consider a beaker or tank of slurry containing particles so dispersed that they can settle independently of one another under the influence of gravity. Originally all particle sizes are uniformly distributed throughout the suspension. All particles start immediately to settle towards the bottom at rates determined by their size, shape, and density—among other factors. Particles reaching the bottom settle out of suspension. After some given time the slurry above the deposited solids is siphoned off or otherwise removed, leaving the deposited solids behind. The deposited solids, together with entrained and any unremoved supernatant slurry, represent the underflow of a classifier. The slurry removed represents the overflow.

There is a certain *separation* size of particle which during the sedimentation has just time to settle from the top surface of the slurry to the level at which underflow and overflow are parted. All particles with a faster settling rate will have dropped to below this level, no matter where in the beaker they started from originally, and therefore will be absent from the overflow. Particles with a slower settling rate will not reach this level if they started from the top of the slurry, but a portion of them,

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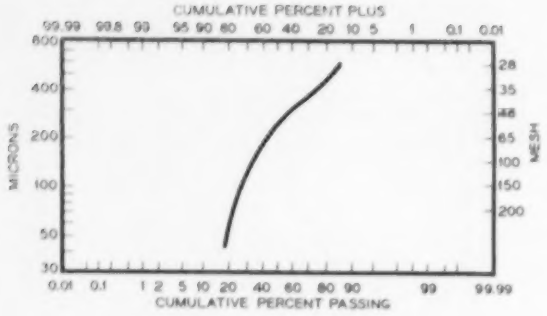

Fig. 1—Example of screen analysis plot of feed.

starting lower in the beaker, will reach the underflow fraction in the time allowed. Since the distance settled in a given sedimentation time by any group of particles having identical settling rates is directly proportional to their settling rate, and since they are originally uniformly dispersed throughout the suspension, it follows that the fraction of any group capable of settling into the underflow is numerically:

$$F = \frac{\text{(settling rate of particle group in question)}}{\text{(settling rate of "separation" particles)}} \quad [1]$$

where

F = fraction of undersize particle group which settles into underflow.

In addition to particles that travelled into the underflow fraction by settling, there is a certain amount of slurry that occupied the underflow space originally. A part of this will be displaced by the settling solids, but some also will remain to fill the void spaces between the settled solids. This void-filling carries into the underflow a certain amount of unclassified feed solids.

Underflow from a batch settling pool, then, comprises three different classes of particles: 1) the oversize, or particles having settling rates sufficient so that all are collected in the underflow; 2) *critical sizes* or undersize particles of which part settle into the underflow; and 3) the void-filling, or material which arrived in the underflow not by settling, but by flowing there as part of the original feed slurry.

It will now be apparent that the underflow from the batch settling pool described above will not cut off sharply below the separation size, but must contain at least some particles of every settling rate present in the original feed. The overflow, on the other hand, would be free of particles that are *oversize* as judged by their settling rate.

Classification products, however, are not judged by their settling rates. They are judged by screen analyses. If the particles were all spheres of uniform density there would be no discrepancy, but particles of nonuniform density or shape will be sorted differently by a screen than by a sedimentation process.

It is obvious that particles of greater density will settle faster, size and shape being identical. When comparing an assembly of objects of nonuniform shape, however, the exact meaning of *size* has been shown equivocal.[3] There is no reason why classification according to settling rate of uniform density particles may not be at least as good a classification according to *size* as screen separation, especially if the *size* classification is judged on the basis of the process purpose it serves. Screen analysis has nevertheless been accepted, perhaps tacitly, as an arbitrary standard of particle sizing. A group of particles having identical settling rates and density but of nonuniform shape will include a more or less narrow band of sizes if judged by screen analysis. Therefore even perfect classification according to settling rate would constitute something different from perfectly sharp classification on the basis of screen analysis. This discrepancy contributes to the lack of sharpness apparent in settling pool classification.

Continuous Settling Pools Produce Same Separation as Batch Pools: The same mechanisms described for batch pools apply also to continuous ones. This is obvious if a somewhat idealized continuous settling pool or desliming pond as sometimes presented in textbooks is considered. Feed slurry is considered to enter one side of such a pond and to flow uniformly across it. Any cylindrical element of feed pulp extending from top to bottom of the pond may now be considered as it traverses the pond. The element may be treated as a little batch pool. It is formed at the input side of the pond from feed slurry with uniformly dispersed solids. Particles settle during the time it is traversing the pond, and the overflow fraction is separated off at the output side of the pond. The classification produced would be identical to that obtained in a batch pool.

It is not so obvious that practical classifiers should give the same results as the idealized pond described above. In actual classifiers the idealized flow pattern does not necessarily exit. Infeed turbulence and that produced by the solids-raking mechanism would be expected to cause eddying and mixing of elements. Also it is not immediately apparent why some elements should not reach the overflow by a path that is in effect shorter than that taken by others, so that classification would take place at different sizes in different elements. There is, fortunately, a self-stabilizing effect which works to minimize mixing and short-circuiting.

Classification Stabilized by Specific Gravity Stratification: During the settling period a pulp density gradient develops in any batch pool or element. The pulp at the top, being soonest depleted of any given size of particles, will have at any time a lower percentage of solids and thus a lower specific gravity than pulp lower in the pool. This is the basis for several of the sedimentation methods for particle size analysis. Hydraulically a pulp tends to act much like a homogeneous fluid, and a lighter fluid tends hydrostatically to remain always in a stratum above a heavier fluid. A significant amount of energy would have to be added to the system to lift

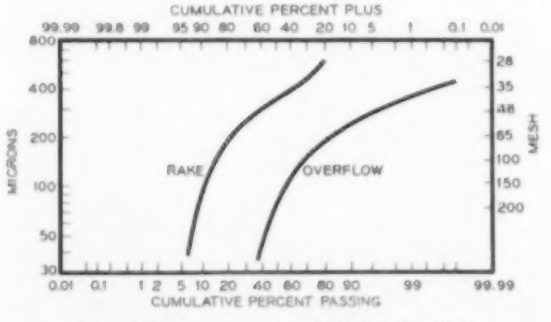

Fig. 2—Example of screen analysis plot of product.

the higher gravity layers and remix them with the supernatant lighter ones. Furthermore, if any disturbance short of an intimate mixing is introduced, gravity and the density differentiation will tend to return the system subsequent to its predisturbance condition. Thus, practically, a pool in which a significant density differentiation and stratification has developed will withstand an astonishing amount of disturbance before the sharpness of classification is noticeably affected.

In a continuous pool also there is a density stratification from top to bottom. At the very bottom of commercial units, such as raking classifiers, there is settled material of high percent solids and low mobility which the rakes push up the deck. Above this the pulp density ordinarily drops off quite rapidly.

Table I. Screen Analysis of Feed to Classifier. Illustrative Problem Calculated on Page 116

Mesh (Tyler)	Cumulative Percent Plus
28	13.7
35	24.0
48	42.3
65	56.5
100	67.2
150	73.3
200	77.9

The incoming feed, having a pulp specific gravity substantially higher than that of the overflow pulp layer at the surface of the pool, plunges immediately below it and spreads out at its level of hydrostatic equilibrium just above the settled solids, or above higher density layers if these exist. While it is spreading out horizontally classification takes place. Solids plus any entrained feed pulp destined for the underflow drop out. The residual pulp, now lightened through loss of underflow solids, is displaced upward by the continuing invasion of new feed pulp.

The rate of upward displacement of the overflow fraction is enough to carry upwards to the overflow and thus out of the classifier all undersize particles remaining in the pulp after the classification. This follows from the *area principle* described in textbooks.[5] The same principle teaches that the thickness of the strata in which the feed moves with a horizontal component does not affect the size at which classification is made.

If there were any lack of uniformity in the classification, the overflow fraction from elements having a finer classification point would be of lower percent solids than that from others and would hence be lower in specific gravity. Since lower gravity pulps tend always to stratify above higher gravity ones, the lower gravity elements would be displaced upwards preferentially, decreasing the effective sedimentation time for successive elements following the same path, thus coarsening their separation and correcting the nonuniformity.

Classification in settling pools, then, is stabilized and made uniform through the effects of specific gravity differentiation and stratification. Unless agitation is excessive or improperly applied, the classification result will be very nearly the same as that obtained in batch pools, and may be calculated on the basis of the batch model.

Calculations of Separation Sharpness—Uniform Density Solids: Classification calculations for dispersed solids of uniform density are considerably simpler than those for solids containing a mixture of materials. Therefore they will be presented first.

In principle, the calculations consist of analyzing mathematically some specified quantity of feed solids into a series of size bands or *partials*, each of which covers a limited range of the particle sizes

Notations Used in This Article

D = dilution of feed pulp = $\frac{(100 - \text{pct solids in feed})}{(\text{pct solids in feed})}$

d = diameter of particle.

F = fraction of undersize band that settles out of suspension.

m = particle size in microns at which mineral classification takes place in a heterogeneous pulp.

Q = grams of solids remaining in suspension after sedimentation on pulp containing 100 g of feed solids. Total of column **11**, Table II.

s = particle size in microns at which classification takes place in homogeneous pulps, or at which gangue classification takes place in heterogeneous pulps.

U = grams of solids in total underflow resulting from classification of 100 g of feed solids.

V = volume dilution of underflow = $\frac{\text{volume of underflow liquid}}{\text{volume of underflow solids}}$

v = settling velocity of particles.

W = grams of solids which settle out of suspension out of 100 g feed solids = total of column **6**, Table II.

δ = density of solids.

ρ = density of liquid.

Table II. Analysis of Classifier Operation. Illustrative Example Set Up on Classification Calculation Sheet

1	2	3	4	5	6	7	8	9	10	11	12	13
		Feed					Rake Product Calculated			Overflow Calculated		
Relative Size Band	Diam., μ	Cumulative, Pct	Grams in Band	F	Grams Settled (4x5)	Grams Overflow in Voids	Total Grams Rake (6 + 7)	Individual, Pct	Cumulative, Pct	Grams	Individual, Pct	Cumulative, Pct
1.000s	347	33.8	33.8		33.2	0.2	33.4	50.3	50.3	0.6	1.3	1.3
0.707s	245	50.1	16.3	0.76	12.7	0.9	13.6	20.4	70.7	3.6	8.1	9.4
0.500s	173	62.4	12.3	0.47	5.8	1.6	7.4	11.1	81.8	6.5	14.5	23.9
0.353s	123	70.4	8.0	0.28	2.2	1.5	3.7	5.5	87.3	5.8	13.0	36.9
0.25s	87	75.7	5.3	0.17	0.9	1.1	2.0	3.0	90.3	4.4	9.8	46.7
0.18s	62	79.2	3.5	0.09	0.3	0.8	1.1	1.7	92.0	3.2	7.1	53.8
0.13s	44	81.9	2.7	0.05	0.1	0.7	0.8	1.2	93.2	2.6	5.8	59.6
Passing			18.1		0.0	4.5	4.5	6.8		18.1	40.4	
Totals					55.2	11.3	66.5			44.8		

present. Average settling rate of particles in each band relative to that of the separation size, s, is estimated on the basis of settling laws or particle dynamics. This determines the fraction of each size band which would settle out (Eq. 1) and hence permits calculation of the actual quantity of each partial which would settle from the specified quantity of feed slurry solids.

In accord with the batch model discussed above, the void-filling would be taken into account by calculating addition of enough more unclassified feed slurry to the settled solids to give the expected dilution to the underflow product. In practice it is mathematically convenient to calculate instead the addition of an appropriate amount of overflow slurry to the settled solids. As carried out the calculations are equivalent, since adding a certain amount of overflow slurry to the settled solids would give the same results as first mixing this amount of overflow slurry with enough of the settled solids to reconstitute feed slurry, and then adding this feed slurry to the residual solids.

From the quantity of settled solids plus void-filling solids determined for each size band, the underflow composition can be calculated. The overflow composition can be calculated from the amount of solids not settling out in each band.

An Example of Detailed Calculation Procedure

Problem: A slurry of solids having a screen analysis as given in Table I is to be classified at a diameter of separation, s, of 347μ in a rake classifier. Specific gravity of the solids is 2.65, and they are suspended in water at 52.0 pct solids. It is desired to calculate the screen analyses of the products.

Procedure: 100 g of feed solids are used as a basis.

1) Plot screen analysis—particle diameter (screen opening) in microns vs cumulative percent plus the diameter, as shown in Fig. 1. This permits choosing the proper size range bands for the calculations. A standard log-probability plot as shown in Fig. 1 displays normal screen analysis data conveniently.

2) Using a layout as shown in Table II, fill in column 1 with the relative size band boundary sizes as shown. Particles finer than the separation size are divided into a series of six adjacent size band boundary sizes, each having a lower size limit which is $\sqrt{1/2}$ times the upper size limit. A seventh band includes all material finer than the sixth.

3) Fill in column 2 by writing the separation size, s, (347μ in the example) on the first line, and on the following lines the micron sizes obtained by multiplying s times the fractional numbers in the first column. This determines the actual size boundaries of the size bands.

4) Fill in column 3 with the cumulative percent plus the micron sizes shown in column 2, obtaining this data from the screen analysis plot shown in Fig. 1.

5) Fill in column 4 by deducting from each figure in column 3 the one immediately above it. This gives the percent of the total feed solids contained in each of the size bands. Since 100 g of solids were used as a basis, it is equal to the actual grams of solids in each size band.

6) Insert factors F in column 5, obtaining these from the most nearly appropriate column in Table III or Table IV, as indicated by captions accompanying these tables. (The column for 417-μ separation, Table III, in the example.) The factors, F, are the fractions of the given size bands which may be expected to settle into the underflow. The derivation of these factors will be discussed in a subsequent section of the report.

7) Fill in column 6 by multiplying the figures in column 4 by the corresponding figures in column 5, excepting the first line (the plus s fraction). To fill in the first space in column 6, deduct 0.6 g from the corresponding figure in column 4.

The operation multiplies the grams solids in each size band times the fraction of the solids that settle, thus giving the grams that settle. The deduction of 0.6 g from the oversize, which according to the model should all settle, is an empirical correction. It takes care approximately of the particle shape variation and of whatever deviation does exist between batch and continuous operation. In calculating batch classification, 0.3 g would be subtracted rather than the 0.6 g specified above.

8) Sum up the figures in column 6. This gives W, the total grams of solids that settle per 100 g of original feed.

9) Fill in column 11 by deducting the figures in column 6 from those in column 4. This gives the grams of solids in each size band which do not settle but remain in suspension to form the gross overflow.

10) Sum up the figures in column 11. This gives Q, the total grams of solids in the gross overflow per 100 g of feed solids.

11) Use the data in column 11 to calculate the size analysis, on individual and cumulative basis, of the overflow and fill in columns 12 and 13 respectively.

12) Calculate the fraction of gross overflow solids (column 11) that go to the underflow as void-filling by the formula:

$$\text{Fraction of gross overflow to voids} = \frac{\dfrac{W}{100\delta D}}{\dfrac{V}{\rho}} - Q \qquad [2]$$

In the example $W = 55.2$, $\delta = 2.65$, $\rho = 1.00$, $D = 0.923$, $V = 0.93$, and $Q = 44.8$. Hence fraction to voids

$$= \frac{55.2}{\dfrac{92.3 \times 2.65}{0.93}} - 44.8 = 0.253$$

If V is unknown, estimate from Table VII.

Fill in column 7 by multiplying the corresponding figures in column 11 by the fraction of gross overflow to voids as calculated above. This gives the grams of void-filling solids that join the settled solids in each size band to constitute classifier underflow.

13) Fill in column 8 by adding the figures in column 7 to those in column 6. This gives the total grams of solids in each size band going to the classifier underflow.

14) Use the data of column 8 to calculate the size analyses on individual and cumulative percent basis of the underflow or rake product, and fill in columns 9 and 10 respectively.

15) Plot the cumulative size analyses from columns 10 and 13 as shown in Fig. 2. This permits converting size analyses to standard screen analyses, which can be read from the plots.

16) Calculate percent solids in overflow.

$$\text{Percent solids in overflow} = \frac{100\,(Q)}{100\,D + Q} \qquad [3]$$

$$= \text{(example)}\ \frac{100 \times 44.8}{(92.3) + (44.8)} = 32.7 \text{ pct}$$

The classification calculated by way of example was actually being carried out in a 6 x 23-ft long DSF Dorr classifier operating at 19 strokes per min. Weir depth was 40 in. and the slope 2½ in. per ft.

Table III. Settling Factors* for Normal Conditions

Size Range of Undersize,† Diam Fraction	Mesh Diam, Mm	28 0.589	35 0.417	48 0.295	65 0.208	100 0.147	150 0.104	200 0.074
—1.000 + 0.707		0.780	0.780	0.780	0.780	0.780	0.750	0.722
—0.707 + 0.500		0.465	0.465	0.465	0.465	0.450	0.380	0.361
—0.500 + 0.353		0.275	0.275	0.275	0.268	0.226	0.190	0.181
—0.353 + 0.25		0.165	0.165	0.158	0.134	0.113	0.095	0.091
—0.25 + 0.18		0.098	0.094	0.079	0.067	0.056	0.048	0.046
—0.18 + 0.13		0.056	0.047	0.040	0.033	0.028	0.024	0.023

* Each factor expresses the number of grams of material, in the size range shown in the first column, which settle out of 1g of material of the same size range originally present in the feed.

† Figures in this column are size ranges, expressed as fractions of the diameter at which the separation is made, of the various portions of the undersize.

Table V shows the comparison between the calculated results and the actual plant operating data. Table VI shows the comparison between plant or observed data and calculated results for a number of operations. Agreement generally is close.

Table IV. Settling Factors* for Sand Work, Slime-Free Pulps, and High Dilution

Size Range of Undersize,† Diam Fraction	Mesh Diam, Mm	28 0.589	35 0.417	48 0.295	65 0.208	100 0.147	150 0.104	200 0.074
—1.000 + 0.707		0.82	0.82	0.82	0.80	0.78	0.75	0.72
—0.707 + 0.500		0.56	0.55	0.51	0.48	0.43	0.38	0.36
—0.500 + 0.353		0.37	0.35	0.30	0.26	0.22	0.19	0.18
—0.353 + 0.25		0.23	0.21	0.16	0.13	0.11	0.095	0.091
—0.25 + 0.18		0.14	0.11	0.085	0.067	0.055	0.050	0.046
—0.18 + 0.13		0.074	0.057	0.045	0.035	0.028	0.025	0.023

* † See corresponding footnotes in Table III.

Factors calculated from the correct free settling

Make separate layout sheets such as shown in Table II for gangue and

Effect of Thickening: In the preceding calculation it has been assumed that the solids are *fully dispersed* and hence have no tendency to cohere into a structure that locks particles of different sizes together. Where such a structure is formed, the solids are constrained to settle at identical rates. They merely *thicken* on settling, rather than classifying or *hydroseparating*.

In most metallurgical pulps at low dilution the solids cohere and hence form a nonsegregating or thickening structure. As the pulp is made more dilute, however, the strength of this cohesion is lessened. Larger, heavier particles are then able to break through the plastic structure of the pulp and settle independently, even though the forces of cohesion among the particles may still be great enough to constrain the smaller ones.

If classification is to take place, it will be obvious that the pulp must be dilute enough, or otherwise conditioned, so that at least oversize particles can segregate or settle independently through the pulp.

Table V. Comparison of Calculated Results with Operating Results. Example Problem

Screen Analysis Feed		Screen Analysis Rake		Screen Analysis Overflow	
		Calculated	Observed	Calculated	Observed
Mesh (Tyler)					
28	13.7	20.6	20.5		
35	24.0	35.9	35.8	0.25	0.3
48	42.3	61.0	61.6	3.9	3.8
65	56.5	77.0	76.7	16.0	16.2
100	67.2	85.0	85.2	31.0	31.3
150	73.3	88.8	89.0	41.8	42.0
200	77.9	91.2	91.6	50.2	50.5
Solids, Pct	52.0	½	74.0	32.7	32.6
Weights	100	66.5	66.7	33.5	33.3

Table VI. Comparison of Calculated and Observed Results Screen Analyses, Percent Plus Mesh

Case I	Gold Ore		Raking Classifier 3 Ft wide x 18 Ft 4 In. Long	
	Rake Product		Overflow	
Mesh	Observed	Calculated	Observed	Calculated
+35	48.0	48	0.46	0.2
+48	66.8	67.0	3.0	2.6
+65	79.6	79.5	11.4	12.1
+100	86.6	86.5	26.0	26.0
+150	90.0	90.0	37.8	37.3
+200	92.2	92.2	49.3	47.4
Total feed solids, pct	27.3	27.2	72.7	72.8

Case II	Copper Ore		Raking Classifier 8 Ft Wide x 32 Ft Long	
	Rake Product		Overflow	
Mesh	Observed	Calculated	Observed	Calculated
+100	76.9	77.5	3.0	3.3
+150	88.5	89.8	16.5	16.0
+200	94.0	94.5	31.0	31.0
Total feed solids, pct	77.9	76.8	22.1	23.2
Solids, pct	75.0	—	13.9	14.3

Case III	Gold Ore		Experimental Batch Settling and Decantation	
	Rake Product		Overflow	
Micron Size	Observed	Calculated	Observed	Calculated
+525	55.4	55.4	0.7	0.7
+371	70.5	69.8	5.8	6.7
+263	81.0	80.1	19.0	20.8
+185	87.6	86.7	36.2	37.2
+131	91.2	90.9	51.2	51.1
+95	93.2	93.0	60.0	60.1

Case IV	Gold Ore		Spiral Classifier, 60 In.	
	Rake Product		Overflow	
Mesh	Observed	Calculated	Observed	Calculated
35	31.6	33	0.1	
48	54.1	54	1.3	
65	72.5	74	9.4	9
100	83.6	83.5	21.2	22
150	88.4	89	35.4	35
200	91.5	92	46.9	46
Total feed solids, pct	77.5	77.7		

Table VII. Approximate Normal Volume Dilution of Classifier Rake Products

Mesh of Separation	V = Volume Dilution of Underflow
1. Raking Classifiers: Solids Raked Up Sloping Deck	
28	0.67 to 0.75
35	0.75 to 0.9
48	0.85 to 0.93
65	0.88
100	0.93
150	0.98
200	1.03
2. Hydroseparators: Spigot Discharge of Machine	
All separations	1.5 to 4.0

Practically, the pulp is usually diluted so that even particles much finer than the mesh of separation are released also. In many cases, however, the extreme fines thicken in the classification pool, leaving a supernatant that is essentially clear water, rather than a suspension graded with respect to particle size as described for the ideal case. In such cases the *void-filling* will consist of thickened pulp, and hence the underflow product will be somewhat dirtier than calculated.

Settling Factors

The settling factors given for the various size bands in Table III have proved themselves reasonably valid for ordinary slimy metallurgical pulps. They were calculated from the following relationships:

For 200 mesh and finer, in a given pulp,

$$v \propto d^2 \quad \text{(Stokes' relationship)}$$

from whence it follows

$$\frac{v_1}{v_s} = F_1 = \left(\frac{d_1}{s}\right)^2 \qquad [4]$$

The mean particle size in any size band was assumed to have the average settling rate for the band. Thus for the band between s and $\frac{s}{\sqrt{2}}$ ($=0.707s$), the mean size or d_1 is 0.85. The value of F_1 for this band is therefore

$$F_1 = \left(\frac{0.85}{1}\right)^2 = 0.72.$$

For coarser than 200 mesh particles, the settling factors of Table III were calculated from the approximate relationship $v \propto d^{3/2}$. Actually this latter formula does not give the theoretically true relationship for free settling spheres, but the factors thus calculated have been validated for ordinary slimy ores through innumerable calculations. Therefore they are considered more accurate under the conditions existing in an ordinary slimy metallurgical pulp than factors calculated on the basis of free settling spheres.

Factors calculated from the correct free settling velocities of quartz spheres in water, as determined by Dorr and Roberts,[5] are given in Table IV. These are recommended for use with slime-free pulps.

Practically, the factors given in Table III and IV may be used directly for all normal metallurgical solids without serious error. Strictly speaking, each column of factors is valid only for solids of 2.65 sp gr. For theoretically precise work, new factors should be calculated for solids having densities substantially different from 2.65, on the basis of the Reynolds Number range on which they are settling.

Separation Sharpness—Heterogenous Pulps

In most classification problems the solids are either of uniform density or of a uniform density gangue with a relatively small fraction of heavier mineral having negligible effect on the screen analysis of the products. At times, however, it becomes desirable to calculate the classification to be expected in pulps containing two or more different kinds of solids of different densities. This can be done following the principles outlined, making a separate calculation for each density class. The procedure will not be described in detail but will be outlined for the case of a pulp containing a substantial amount of heavy mineral accompanied by a lighter gangue.

First, the distribution of mineral and gangue in each screen partial must be determined. Next make an analysis graph in which particle diameter (screen opening) is plotted against the cumulative grams of gangue coarser than this diameter which would be present in 100 g of *total* feed solids (gangue plus mineral). On the same graph make a similar plot for the mineral. The sum of these two curves at any mesh gives the total percent solids plus the mesh. These plots make it possible to determine the number of grams of gangue and of mineral in any size band which would be present in 100 g of total solids.

Choose a separation size, *s*, for the gangue. Estimate the size of mineral particles, *m*, settling at the same rate on the basis of *settling ratio* as described in such text books as Taggart's.[5]

Make separate layout sheets such as shown in Table II for gangue and for mineral, the gangue being calculated for the gangue separation *s* and the mineral for a separation at *m*. Proceed as for two separate homogeneous pulps except as noted below:

1) The cumulative grams plus any given size per 100 g *total* feed solids (column **3**) is read for each material from the respective plots on the analysis graph described above.

Note that the total of column **4** on the gangue sheet equals the percent gangue in the feed solids; the total of column **4** on the mineral sheet equals the percent mineral in the feed solids.

2) To fill in the first spaces in column **6** (step **7**) deduct a proportionate share of the 0.6 g correction in each sheet. That is, for the gangue sheet deduct $(0.6)\frac{(\text{percent gangue in feed})}{100}$. For the mineral sheet deduct $(0.6)\frac{(\text{percent mineral in feed})}{100}$.

3) All percentages (columns **9**, **10**, **12**, **13**) are based on combined total weights for both sheets (gangue + mineral). Thus the figures in column **12** are obtained by multiplying corresponding figures in column **11** by the factor $\frac{100}{Q_{mineral} + Q_{gangue}}$. The figures in column **9** are obtained by mutiplying the corresponding figures in column **8** by the factor $\frac{100}{U_{mineral} + U_{gangue}}$.

Note that the total of column **12** on the gangue sheet equals the percent gangue in the overflow solids; the sum of column **12** on the mineral sheet equals the percent mineral in the overflow solids. Likewise the total of column **9** on the gangue sheet equals the percent gangue in the underflow and that on the mineral sheet the percent mineral in the underflow, where U = total of column **8** and Q = total of column **11**.

4) The void-solids are calculated as follows (step 12, Eq. 5):

$$\text{Fraction of gross overflow to void} = \frac{\frac{W\ \text{gangue}}{\partial\ \text{gangue}} + \frac{W\ \text{mineral}}{\partial\ \text{mineral}}}{\frac{100D}{V\rho} - \left(\frac{Q\ \text{gangue}}{\partial\ \text{gangue}} + \frac{Q\ \text{mineral}}{\partial\ \text{mineral}}\right)} \qquad [5]$$

5) Plot the cumulative percent plus vs size of mesh for both overflow gangue and overflow mineral (column **13**). (This corresponds to step **15**.) Plot the sum of the percents plus for the two curves to obtain completed screen analysis of overflow solids. Do the same for the underflow analyses (column **10**).

6) Overflow percent solids (step 12, Eq. 6).

$$\text{Percent solids overflow} = \frac{100\ (Q_{mineral} + Q_{gangue})}{100\ D + Q_{mineral} + Q_{gangue}} \qquad [6]$$

Part II. A Method for Calculating Countercurrent Classification

As shown in Part I, the sharpness of classification which can be produced in any simple sedimentation-type classifier is limited. The underflow always contains a certain fraction of the fines originally present in the feed. These fines arrive in the underflow both by partial settling out of *critical-sized* material and by entrainment of overflow pulps as *void-filling* in the underflow.

Overall separations can be sharpened by repulping the underflow in clean water and reclassifying. Repeated reclassification of underflow in clean water theoretically will serve to sharpen the overall separation to the maximum possible by hydraulic means, and this fact is utilized in the decantation methods of size analysis.

Repeated repulping of underflow in clean water for reclassification through a series of machines has the disadvantage of requiring a lot of clean water. The combined, fine-bearing overflow slurry will then often be excessively dilute. This particular disadvantage can be largely overcome, while sharpened separation is still obtained, through feedback of the classifier overflows from stage to stage in a countercurrent decantation sequence. A method for calculating the products to be expected theoretically from countercurrent classification in a series of sedimentation-type units is presented in this memorandum. The calculated and observed results cannot be expected to correspond as closely as they do for single-stage classification. First, errors in estimation of settling rates may be cumulative through a countercurrent sequence of several classifications. Second, the sharper the overall classification produced, the more apparent and disturbing the effects of nonuniformities in particle shape or density will be. However, the calculations should serve to show approximately what might be expected from countercurrent classification.

Calculations are carried out on the following principle: A specified quantity (100 g) of feed solids is analyzed mathematically into a series of adjacent particle-size bands or partials. A material balance is made for each of these partials through the countercurrent classification flowsheet to be calculated. This permits determination of the amount of each partial which appears in the underflow from the final classifier in the series, and the amount carried in the overflow from the system. The screen analyses of final underflow and ultimate overflow solids can then be constructed.

Table VIII. Values of M when S = 0.2

Band	Relative Size	R	M_1	M_2	M_3
1	−1, +0.841	0.139	0.878	0.865	0.861
2	−0.841, +0.707	0.47	0.679	0.591	0.557
3	−0.707, +0.597	0.86	0.539	0.385	0.309
4	−0.597, +0.500	1.28	0.44	0.255	0.166
5	−0.500, +0.422	1.7	0.352	0.179	0.091
6	−0.422, +0.354	2.12	0.32	0.132	0.058
7	−0.354, +0.298	2.51	0.285	0.102	0.039
8	−0.298, +0.25	2.85	0.260	0.083	0.029
9	−0.25, +0.21	3.13	0.242	0.073	0.023
10	−0.21, +0.177	3.35	0.230	0.064	0.019
11	−0.177, +0.149	3.5	0.222	0.060	0.017
12	−0.149, +0.125	3.65	0.215	0.056	0.015
13	−0.125, +0	3.85	0.206	0.051	0.013

Table IX. Values of M when S = 0.3

Band	Relative Size	R	M_1	M_2	M_3
1	−1, +0.841	0.116	0.894	0.883	0.882
2	−0.841, +0.707	0.39	0.720	0.65	0.625
3	−0.707, +0.597	0.69	0.597	0.467	0.406
4	−0.597, +0.500	0.96	0.51	0.347	0.266
5	−0.500, +0.422	1.23	0.448	0.268	0.179
6	−0.422, +0.354	1.47	0.405	0.216	0.128
7	−0.354, +0.298	1.75	0.364	0.173	0.090
8	−0.298, +0.25	1.84	0.352	0.160	0.080
9	−0.25, +0.21	1.97	0.337	0.146	0.069
10	−0.21, +0.177	2.06	0.327	0.137	0.066
11	−0.177, +0.149	2.13	0.319	0.130	0.058
12	−0.149, +0.125	2.19	0.313	0.125	0.054
13	−0.125, +0	2.3	0.303	0.116	0.048

Distribution of each partial in the countercurrent classification series may be solved by the x method of material balance, illustrated in Fig. 3. To carry out such a calculation, it is necessary to know in each stage, for each class or size-range of particles, the corresponding value of R, which is defined as the fraction of the class reporting to the classifier overflow, divided by the fraction of the class reporting to the classifier underflow. The value of R is derived as follows:

In any stage of sedimentation classification the amount of any given size band that appears in the underflow is

$$B = F + (1 - F)S$$

where B = fraction of size band in underflow; F = settling factor, or fraction of size band settling into underflow; S = fraction of feed water to underflow (= fraction of *gross overflow* to underflow).
Fraction to overflow = $1 - B$.
Therefore:

$$\frac{\text{(Fraction to overflow)}}{\text{(Fraction to underflow)}} = R = \frac{1 - B}{B} = \frac{(\quad 1 \quad)}{(F + S - SF)} - 1 \qquad [7]$$

The settling factor, F, for each partial is found as described in Part I, and S for each stage may be determined from a water balance. The appropriate values for R can then be calculated, and the distribution of each partial solved by the x method. Usually it will be necessary first to use a tentative or trial water balance to obtain approximate values of S and thence of R. After then completing an approximate balance of all classes of solids, a more accurate or second approximate water balance can be made and the distributions recalculated. By successive approximations, the desired degree of accuracy can be attained.

In almost all real systems, the value of S and hence of R for any partial will be different in different stages. However, some idea of the effectiveness and characteristics of simple countercurrent classification can be obtained from considerations of hypothetical systems in which the value for R remains constant in all stages.

First there is a limitation to the effectiveness of simple countercurrent decantation for classification.

Where the value of R remains constant through a series of countercurrent stages, the fraction of any

Calculation of Countercurrent Classification Overall Factors by X Method

x = quantity (unknown) of solids within the specified size range appearing in the underflow from stage n.
R_n =

Fraction of these solids to overflow in stage n

Fraction of these solids to underflow in stage n

The quantity of solids fed into any stage n from a preceding stage n-1 is equal to $[(R_n)$ (solids in underflow from stage n) $+ x]$.
Solving progressively from final stage,

$$Jx = R_n x + x \text{ or } J = R_n + 1$$
$$K = JR_{n-1} + 1$$
$$L = KR_{n-2} + 1$$

and $1/L$ = fraction of solids originally advancing into stage $(n - 2)$, which finally appear in underflow from final stage (n) = overall factor, M_n for three stage countercurrent classification.

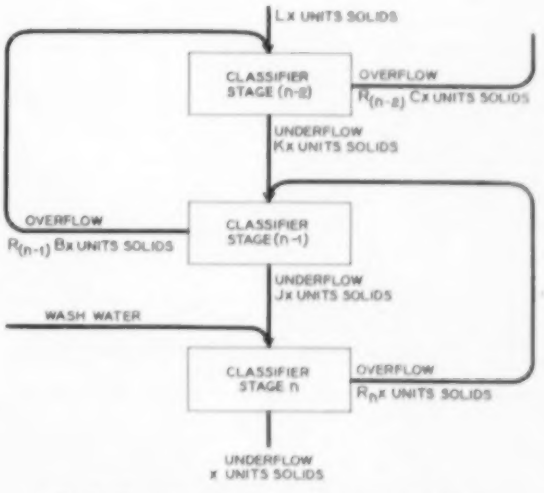

Fig. 3—Illustration of the X method of material balance.

feed partial appearing in the final underflow, M, can be shown to be

$$M = \frac{1 - R}{1 - R^{n+1}}. \qquad [8]$$

Where R is less than 1, and $n \to \infty$, the value of M approaches as an asymptote

$$M \to 1 - R \qquad \text{where } R < 1$$
$$n \to \infty$$

Therefore no matter how many stages of simple countercurrent classification are used, the overall factor for any partial for which R is less than 1 will approach $1 - R$ as an asymptote. Elimination of such partials cannot be improved indefinitely.

Tables VIII and IX show the fraction M of specified size bands which would appear on the final underflow of countercurrent classification series of one, two, and three stages (one stage is obviously simple classification), for solids settling according to Stokes' law (to determine settling factors) and for specified values of S (which is taken as constant for all stages). The upper boundary of each band is $\sqrt[4]{2}$ coarser than the lower boundary rather than $\sqrt{2}$ as used in the example of Part 1. Narrower bands are used in the present calculations in order to follow more precisely the sharper classifications expected. Thus it takes two of the bands here shown to make full *critical*.

It is immediately apparent, in comparing the recovery values M for one stage and three stages, that simple countercurrent classification is not remarkably effective in improving elimination of the first critical (bands 1 and 2). After the value of R reaches unity (essentially at band 4) the improvement obtained by countercurrent classification becomes substantial, and for finer particles the eliminations become very high indeed.

Compound Countercurrent Classification

Elimination of the first few criticals can be increased spectacularly if the stages of countercurrent classifications are adjusted to progressively coarser meshes of separation. This is obtained at the expense of an increase in the internal circulating loads within the system, but if it is judiciously used the circulating loads need not become excessive, while the improvement in classification may be marked.

Table X. Compound Countercurrent Classification for Two Stages*

Band		M	First Stage Load, Pct†	Second Stage Load, Pct†
−2,	+1.68	1	114	114
	+1.41	1	147	147
	+1.19	1	186	186
	+1.00	1	238	238
−1,	+0.841	0.73	234	197
	+0.707	0.45	186	126
	+0.595	0.25	163	87
	+0.500	0.165	149	65
	+0.422	0.125	139	52
	+0.354	0.098	134	43

* Second stage is two meshes coarser than first, $S = 0.2$. † Percent stage load is defined as the actual quantity of solids within any size band fed to a stage, divided by the quantity of such solids in the new feed to the system.

Table X shows the overall recovery factors, M, for a two-stage countercurrent classification in which the second stage is set for a separation two meshes coarser than the primary stage. Here again it is assumed that the water balance is so adjusted that the value of S for each stage is 0.2. It is also assumed that the particles settle according to Stokes' law.

Table XI. Stokes' Law Values of F and R

Band	Relative Size Range	$\sqrt[4]{2}$ Bands Average	F	R $S = 0.3$	R 0.2	R 0.1
1	1 to 0.841	0.921	0.848	0.118	0.139	0.159
2	0.841 to 0.707	0.774	0.599	0.39	0.47	0.57
3	0.707 to 0.595	0.651	0.424	0.68	0.86	1.07
4	0.595 to 0.500	0.548	0.300	0.96	1.28	1.7
5	0.500 to 0.422	0.461	0.212	1.23	1.7	2.44
6	0.422 to 0.354	0.387	0.150	1.47	2.12	3.26
7	0.354 to 0.298	0.326	0.106	1.75	2.51	4.13
8	0.298 to 0.25	0.274	0.075	1.84	2.95	4.96
9	0.25 to 0.21	0.230	0.053	1.97	3.13	5.75
10	0.21 to 0.177	0.194	0.038	2.06	3.35	6.46
11	0.177 to 0.149	0.163	0.027	2.13	3.5	7.06
12	0.149 to 0.125	0.137	0.019	2.19	3.65	7.54
13	0.125 to 0.0	0.063	0.004	2.3	3.85	8.65

The assumed conditions might be met, for example, when making a 200-mesh separation in bowl classifier or hydroseparator. The example will demonstrate in a general way the nature of results that might be expected in any such system. Calculated elimination of the first system critical (first two bands below mesh of separation) averages 40 pct, while elimination of the second system critical (third and fourth bands below mesh of separation) averages 79 pct. This would not be considered poor in a hydroscillator.

Table XI gives the Stokes' law values for F and R.

Table XII. Compound Countercurrent Classification for Three Stages*

Band		M	First Stage Load, Pct	Second Stage Load, Pct	Third Stage Load, Pct
−2,	+1.68	1	100	114	114
	+1.41	1	100	147	147
	+1.19	1	126	212	186
	+1.00	1	207	335	228
−1,	+0.841	0.69	259	345	185
	+0.707	0.30	219	213	93
	+0.595	0.143	185	135	50
	+0.500	0.077	164	95	30
	+0.422	0.050	151	72	20
	+0.354	0.033	142	57	14

* Second stage one mesh coarser than first. Third stage two meshes coarser than first. $S = 0.2$, all stages.

Table XII shows the overall recovery factors for a three-stage operation in which each stage makes a separation progressively one mesh coarser than the stage before it. Recirculation of partials near the mesh of separation is higher than for the two-stage operation. The recirculations are possibly not too high for certain applications, and the eliminations, at least as calculated, compare to those obtained in a sizer or sorting column classifier.

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Heavy Metals In Stream Sediment Used as Exploration Guides

Fig. 1—The Nash Creek zinc-lead prospect was chosen as an experimental area for this study.

Water analysis for traces of base metals became practical about eight years ago with the development of rapid chemical tests that could be carried out at the field site. Since that time it has been widely used in mineral reconnaissance.

by H. E. Hawkes and Harold Bloom

STREAMS and rivers are the principal channels into which the weathering products of rocks and their contained ores are funneled. The inorganic load of a stream system is a crude sample of all the earth materials in the drainage basin of the stream, and the heavy mineral assemblage of sediments is one of the oldest and most successful guides to certain kinds of bedrock ore in unexplored terrain. Early application of this method was made possible by the ease of separation and identification of heavy minerals in the field with the simplest equipment.

Water analysis for traces of base metals became a practical possibility about eight years ago with the development of rapid chemical tests that could be carried out at the field site.[1, 2] Since that time the water method has been widely used in mineral reconnaissance, and a substantial number of discoveries have been reported. Here again, the techniques developed for this work were relatively simple and could be performed easily at the field site.

Sediment analysis by chemical methods, as distinguished from mineralogical, has received less attention. Some experimental work was done by Lovering and others,[3] but to the authors' knowledge, chemical analysis of sediment samples has not been used in routine exploration programs until very recently. In spite of the obvious promise of this method, it was not pursued vigorously because of the lack of a simple and portable chemical test that could be used conveniently at the field site or in a field camp. Procedures that had been developed for trace analysis of soils all called for a rigorous chemical attack with hot acids—tests more conveniently applied in processing large volumes of routine soil samples at an established field headquarters.

Recently an extremely rapid and simple test was developed by Bloom.[4,5] The extractant used in this procedure is a cold solution of ammonium citrate. Elements measured are the *heavy metals*, principally zinc, lead, and copper. This method differs from the established procedures for soil analysis in that it measures only a small fraction, about 5 pct, of the total metal content of the sample. It has been found, however, that the significant pattern of metals in the sediments of streams below a base-metal deposit can be brought out as well and sometimes better by this simple cold-extraction procedure than by the more tedious methods requiring hot acids.

To understand fully the significance of the relation between the readily extractable and the total

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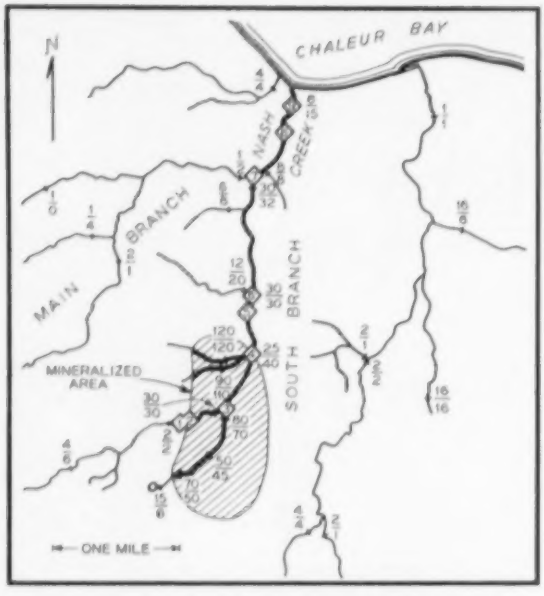

Fig. 2—Within this area samples of both active and floodplain sediments were taken at 23 sites on streams outside the anomalous drainage pattern.

metal content of sediments, it might be helpful to review briefly the various ways in which metals can occur in stream sediments.

Geochemistry of Stream Sediments: Stream beds serve as a temporary resting place for materials in transit between the site of weathering and erosion and the site of ultimate deposition. Solid materials temporarily deposited either in the active channel of the stream or over the floodplain may be grouped as follows:

1) Rock fragments.

2) Primary resistant minerals. These consist principally of quartz and the heavy accessory minerals of igneous and metamorphic rocks, but may include native metals, cassiterite, columbite, and other resistant ore minerals.

3) Secondary minerals formed at the site of weathering of rocks and ores: clay minerals, hydrated oxides of iron and manganese, and the secondary oxidation products of the ore metals.

4) Inorganic matter precipitated from stream water solutions. This may occur as relatively insoluble secondary minerals or as ions absorbed into the structure of minerals already present in the sediments.

5) Inorganic matter withdrawn from ionic solution by living organisms and by the formation of organic complexes.

6) Exchangeable ions in relatively active equilibrium with the aqueous solutions in contact with the sediment.

It should be noted that material in the first three of these groups travels as clastic fragments, whereas that in the last three travels predominantly in solution in the water.

The readiness with which these various kinds of material can be dissolved for chemical determination depends on the kinds of bonds that hold the atoms in place. The primary rock-forming minerals (groups 1 and 2) commonly require a strong chemical attack, such as alkali fluxes or hydrofluoric and perchloric acids; the secondary minerals (groups 3 and 4) are commonly soluble in hot acids; metals held in living organisms and organic complexes (group 5) can be released by ashing or wet oxidation; and exchangeable ions (group 6), with the weakest bonds, will go into solution promptly on treatment with neutral cold solutions of any one of a number of salts, including ammonium citrate. Thus by modifying the extraction treatment, it is possible to get at least a qualitative idea of how a given chemical component is partitioned between the different kinds of material making up the sediment.

Analytical Procedures

Exchangeable Metals: Exchangeable metal, for these purposes, is defined as that fraction of the total metal content of a clastic sample that can be put into aqueous solution by treatment with a cold dilute solution of ammonium citrate. The analytical procedure for determining the exchangeable metal content of soil or sediment samples is described in detail in an earlier paper by Bloom.[4] In essence the procedure is as follows: 1) For the base-camp laboratory, use a dry sample and sieve, discarding the coarse fraction; for use at the field site, collect a moist sample and remove pebbles and fragments of wood by hand. 2) Measure the dry or moist sample into a volumetric scoop of about 0.15-cc capacity and tap into a test tube. 3) Add 3 ml of an aqueous solution containing 5 pct ammonium citrate and 0.8 pct hydroxylamine hydrochloride adjusted to a pH of 8.5. 4) To the same tube, add 1 ml of a 0.001 pct solution of dithizone (diphenylthiocarbazone) in xylene. 5) Shake tube vigorously for 5 sec, observing the color of the dithizone solution that collects at the surface of the aqueous citrate solution. If the color is green, green-blue or blue, record 0, ½, or 1 ml, respectively. If the color is blue-purple to red, titrate with 1 ml or larger increments of dithizone solution, shaking for 3 sec after each addition, until a blue color is obtained. Record the volume of dithizone solution used, as an index of the heavy metal content.

Calibration of these values shows that for zinc, commonly the most abundant of the heavy metals encountered, the conversion factor of milliliters of dithizone solution to micrograms (millionths of a gram) of metal is 0.34. Thus, if 10 ml of dithizone solution is required to reach the blue end point, the sample contains 3.4 mg of exchangeable metal expressed as zinc. If the volume of the scoop is known, the exchangeable metal content of the sample may be computed in absolute values as micrograms per cubic centimeter of sample (parts per million weight/volume, or ppm w/v). Inasmuch as the bulk density of dried samples is commonly about 1, the value of parts per million weight/volume is a fair first approximation of parts per million weight/weight.

The exchangeable metal content of moist, unsieved sediment samples is in general comparable with that in the same samples after systematic drying and coarse sieving. This was shown in a study of a set of 46 samples representing both low and high contents of exchangeable metal. These samples were first analyzed as collected, and then dried, sieved to −12 mesh and reanalyzed. The median of the ratios of exchangeable metal content of the moist sample to that of the same sample after drying and sieving was 1.20.

Total Metals: In the experiments described below, the *total* metal content of the sediment samples was determined by digestion with hot nitric acid for 1 hr following a procedure described in detail by Bloom and Crowe.[6] Attack by nitric acid will remove the metal from virtually all components of the sediment except the primary resistant minerals and rock fragments.

Metals in Water: Water analysis has a unique advantage over sediment analysis in that the component to be determined is already in solution. Thus errors due to inhomogeneities of sample and variations in the rigor of the extraction procedure do not affect the precision of the data. In the field experiments described here, water was analyzed for traces of heavy metals by a method requiring most of the reagents used for the test for exchangeable metals in sediments.

Test for Heavy Metals in Water

Reagents: Water. This refers to metal-free water, usually obtained by passing water through a resin demineralizer.

Dithizone (diphenylthiocarbazone) of reagent grade.

Dithizone in carbon tetrachloride, about 0.01 pct. This is used to purify the sodium acetate solution. Dissolve approximately 0.01 g of dithizone in 100 ml of carbon tetrachloride a few hours before using.

Dithizone in xylene, 0.001 pct, work solution. Prepare a work solution of 0.001 pct by diluting 10 ml of stock solution to 100 ml with xylene.

Carbon tetrachloride, ACS grade.

Xylene, ACS grade. The xylene is further purified in the following manner: 500 ml of xylene are extracted with about 30 ml concentrated sulfuric acid by vigorously shaking for 1 min. Allow phases to separate completely and discard the acid. Add demineralized water to the xylene (ratio 1:10) and mix vigorously. Separate the xylene and distill in an all-pyrex still.

Sodium acetate (anhydrous) solution 8 pct. (The hydrated form of this compound for some unknown reason produces a pH of 7.8 which is not satisfactory). Dissolve 40 g of anhydrous sodium acetate in 500 ml water (pH 6.7). Remove heavy metals by extracting the solution with successive 20 ml portions of the carbon tetrachloride solution of dithizone until the dithizone is no longer red. Wash the aqueous solution with about 3 ml portions of carbon tetrachloride until the carbon tetrachloride is colorless.

Apparatus: Necessary equipment consists of two wash bottles, polyethylene, 8-oz capacity; one glass-stoppered cylinder, pyrex, 50-ml capacity; four reagent bottles, pyrex or polyethylene, 16-oz capacity; and one graduated cylinder, 100-ml capacity.

Procedure: Add the sample of water to the 40-ml mark of the glass-stoppered cylinder, followed by 5 ml of sodium acetate solution. Add ten drops of 0.001 pct dithizone in xylene from a wash bottle, cap the cylinder and shake vigorously for 20 sec. Observe droplets as they collect on the water surface. If purple or red, add ten more drops and shake only 10 sec. If a blue color is not obtained, repeat the latter step until it is obtained. This blue color is known as the end point. As an index of the heavy metal content, record the number of drops required to reach the end point.

A chart for converting drops of dithizone solution to parts per million heavy metal was constructed by connecting the following three points as plotted on graph paper: 0 drops-0 ppm; 40 drops-0.0025 ppm; 150 drops-0.0125 ppm. In normal surface water, zinc is the only member of the heavy metal group present in significant quantities.

Experimental Area

What kind of sediment sample will show a chemical pattern most indicative of mineralization and at the same time minimize the errors due to erratic distribution of metals? This question can be answered only by a study of the distribution of metals in stream sediments below an area of known mineralization, where the drainage has not been contaminated by metal leached from mine workings and ore dumps. The experimental results will be easier to interpret if the patterns in such an area are not confused by more than one source of metal. In other words, the mineralized area should show a relatively sharp boundary with unmineralized terrain.

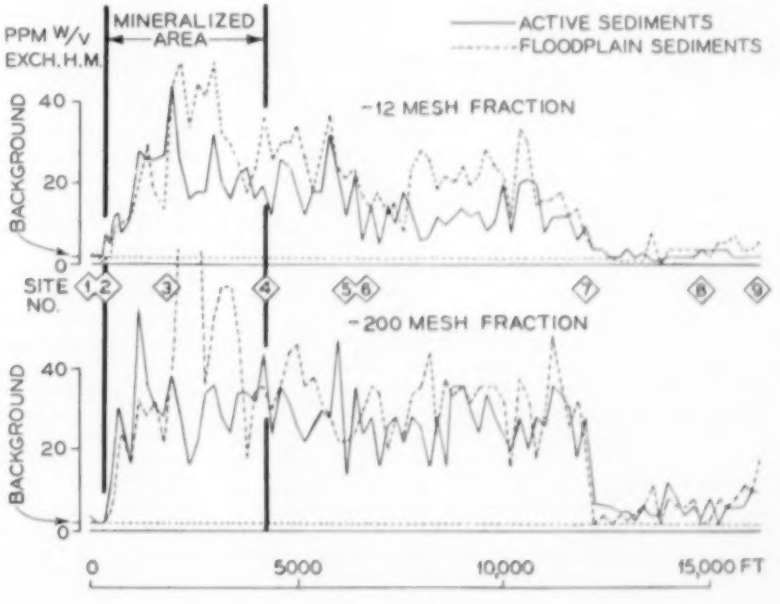

Fig. 3—Normal operating errors where samples are taken without regard to grain size and texture can be estimated by comparing the exchangeable metal content of contiguous samples of active sediment taken at 200-ft intervals along the South Branch of Nash Creek between sites 3 and 7 (solid lines in graph). The error in sampling floodplain sediments was estimated by comparing data on corresponding samples of floodplain sediment (broken lines in graph).

The area chosen for this study was the Nash Creek zinc-lead prospect, about 2 miles south of the town of Nash Creek on the south shores of Chaleur Bay and 25 miles northwest of Bathurst, New Brunswick (see Fig. 1). Subsequent diamond drilling of this prospect has shown that zinc and lead occur as sphalerite and galena replacing altered Devonian volcanics throughout an area of about 1 sq mile. No evidence of copper mineralization has been observed. Calcareous sedimentary rocks are interbedded with the volcanics. A sequence of flat-lying Carboniferous conglomerates and red sandstones lying unconformably over the Devonian complex extends for several thousand feet south of the shore of Chaleur Bay. No other strong geochemical indications of mineralization were found within a radius of several miles of the Nash Creek prospect. There was no contamination, and boundaries of the mineralized area were well defined.

Fig. 2 shows the drainage pattern and location of sample sites in the vicinity. It will be noted that the stream draining the mineralized area, known locally as the South Branch of Nash Creek, is relatively small and is joined by a much larger stream, the Main Branch of Nash Creek, about 6000 ft downstream from the edge of the known mineralized area. The combined drainage of the two branches flows into Chaleur Bay about 3000 ft below this confluence.

Experimental Results: Samples of both active sediments from the present stream bed and of floodplain sediments near the edge of the active channel were collected at 200-ft intervals along the South Branch of Nash Creek from site 1 (see Fig. 2), above the mineralized area, downstream as far as tidewater at site 9. Samples were also taken from other streams within the area of Fig. 2 for the purpose of showing the *background* (or normal) distribution of exchangeable metals in the stream sediments draining unmineralized areas. At selected sites in the drainage below the Nash Creek prospect, suites of experimental samples were taken for special purposes, as described in detail below.

Table I. Active Sediments: Variation in Exchangeable Heavy Metal Content of Repeat Samples from Same Site (—12 Mesh Fraction)

Material	Site	Exchangeable Heavy Metal Ppm W/V*
Ooze	6	21, 25, 28, 30
Ooze	8	5, 7, 8, 10
Silt	5	22, 24, 32, 52
Silt	6	18, 20, 21, 21
Sand	5	12, 14, 16, 17
Gravel	5	16, 16, 17, 18
Gravel	6	8, 11, 12, 12

* Weight/volume: 1 ppm w/v is equal to 1 μ per cu cm.

Background: Within the area of Fig. 2, samples of both active and floodplain sediments were taken at 23 sites on streams outside the anomalous drainage pattern. The median of these values is 2 ppm w/v (equal to micrograms per cubic centimeter) in both the active and floodplain sediments and in both the —12 and —200 mesh fractions. From experience elsewhere, this background for the content of exchangeable metals in sediments appears normal for unmineralized terrain.

The metal content of the stream water was also determined at most of these sites; the median of these values was 0.0006 ppm (0.6 μg/l) of heavy metal expressed as zinc.

Distribution of Metal in Active Sediments: The error in sampling active sediments was investigated by collecting multiple samples of similar material within a few feet of one another and comparing the exchangeable metal contents, as shown in Table I. These results show that if samples of the same kind of material are collected, the sampling error is relatively small.

Table II. Active Sediments: Variation in Metal Content with Sediment Type

Site	Material	Exchangeable Heavy Metal Ppm W/V* —12 Mesh	Exchangeable Heavy Metal Ppm W/V* —200 Mesh	Total Zn Ppm W/W† —12 Mesh	Total Pb Ppm W/W† —200 Mesh
5	Silt	74	60	1000	200
	Sand	20	50	800	100
	Gravel	30	55	600	150
6	Ooze	47	50	500	200
	Silt	37	40	700	200
	Gravel	13	40	700	150
8	Ooze	13	17	125	50
	Silt	13	13	100	50
	Sand	10	27	125	50

* Weight/volume: 1 ppm w/v is equal to 1 μ per cu cm.
† Weight/weight: 1 ppm is equal to 1 μ per gram of sample.

Samples representing two vertical profiles below the active stream bed were collected at site 5. No significant variation with depth was revealed, indicating that for routine work a sample from the surface of the stream bed is as satisfactory as a deeper sample.

The variation of metal content with type of sample was investigated by collecting suites of samples of different kinds of sedimentary material at three sites, 5, 8, and 6, all lying in the anomalous drainage below the Nash Creek prospect. The analytical data, given in Table II, show that the —12 mesh fraction of fine grained samples and of samples with a high content of organic matter are somewhat higher in their content of exchangeable metal than similarly prepared samples of sand or gravel.

Sizing studies were run on a bulk sample of gravel collected at site 5, with results summarized in Table III. Exchangeable metal was determined on weighed samples (ppm w/w) as well as on samples measured in the volumetric scoop (ppm w/v), for the purpose of obtaining more accurate comparison between exchangeable and total metal. These figures show a pronounced inverse relationship between content of exchangeable metal and grain size. The same relationship is borne out by the data of Table II, on samples of different grain size collected from the same sites.

Normal operating errors where samples are taken without regard to grain size and texture may be estimated by comparing the exchangeable metal content of contiguous samples of active sediment taken at 200-ft intervals along the South Branch of Nash Creek between sites 3 and 7 (solid lines of Fig. 3). Of these samples, almost two-thirds were sand and gravel, the remainder silt. In all, 49 pairs of samples 200 ft apart were compared. The median of the ratios of the higher to the lower value for exchangeable metal in these pairs is 1.50 for the —12 fraction, and 1.33 for the —200 mesh fraction.

This means, for example, that if a given sample of active sediment contains 12 ppm of exchangeable metal, a sample taken 200 ft either up or downstream will have a 50 pct chance of falling within the range from 8 to 18 ppm for −12 mesh fraction or 9 to 16 ppm for the −200 fraction.

Distribution of Metals in Floodplain Sediments: In a similar way, the error in sampling floodplain sediments was estimated by comparing data on corresponding samples of floodplain sediment (broken lines of Fig. 3). Here the median of the ratios of the higher to the lower value for exchangeable metal is 1.21. Thus if a given sample of floodplain sediment contains 12 ppm of exchangeable metal, a sample taken 200 ft either up or down stream will have a 50 pct chance of falling within the range from 10 to 15 ppm. The lower sampling error of floodplain sediments as compared with active sediments is unquestionably due in large part to the more uniformly fine grained texture of the floodplain samples.

The relation of the content of exchangeable metals in floodplain material to distance from the active channel of the stream is shown in the data of three traverses crossing the floodplain at sites 5, 6, and 8. The results are summarized in Fig. 4. The data of the first two traverses indicate that in the absence of a lateral source of metal, the exchangeable metal content of floodplain samples is highest near the active channel. Where several active channels are present, as at site 8, the relationship becomes confused.

Samples comparing humus-rich silt with organic muck from the floodplain were collected near site 5; no significant relation between these types of sediment and the content of exchangeable metal was apparent.

Samples representing a vertical profile of the floodplain sediments to a depth of 18 in. were collected at site 6. These determinations showed no significant variation in metal content with depth.

Table III. Active Sediments: Variation in Metal Content with Grain Size, Gravel from Site 5

Mesh Size Minus	Mesh Size Plus	Percent of Total Sample	Exchangeable Heavy Metal: Ppm W/V*	Heavy Metal: Ppm W/W*	Total Zn Ppm W/W*	Ratio Exchangeable Heavy Metal: Total Zn Ppm W/W*
8	32	65	10	6	500	0.01
32	80	25	18	15	500	0.03
80	115	4	24	25	800	0.03
115	200	3	38	45	800	0.05
200		3	55	75	1000	0.07

* See footnotes for Table II.

Ratios of the exchangeable metal content of floodplain samples to that in the corresponding active sediments were computed on the basis of data shown in Fig. 3. For the −12 mesh fraction the median ratio is 1.33, whereas for the −200 mesh fraction it is 1.06. Ordinarily, the active sediment is more coarse grained than the corresponding floodplain sediments, and has a lower content of exchangeable metal. Fine sieving, however, reduces the disparity.

Distribution of Metal Within Source Area: Fig. 3 shows the variation in the metal content of the stream sediments as the stream enters and passes through the source area. At site 1, upstream from the mineralized area, the metal content of the sediments is near the background value. The content of metals increases sharply as the stream enters the source area, at a point commonly known as the *cutoff* (site 2 in Fig. 3). Immediately below this point, the metal content of the active sediment apparently increases faster than that of the floodplain sediments. As the stream flows through the source area, the exchangeable metal content fluctuates in response to local increments from lateral drainage, ground water, and springs. Metal-rich tributary streams are known to enter the main channel at sites 3 and 4, where they can be correlated with local increases in the metal content of the floodplain sediments.

Distribution of Metals Downstream from Source Area: The attenuation of the metal content of sediments downstream from the Nash Creek prospect is, in a very general way, a reflection of simple dilution, as shown by the data of Fig. 3 between site 5 and tidewater at site 9.

At site 7, the South Branch of Nash Creek is joined by the Main Branch, with about four times the drainage area of the South Branch. By a straight dilution factor, the exchangeable metal content of sediments below this point should be 5 and 7 ppm w/v for −12 and −200 mesh fractions respectively. Averages of the values actually observed in the section below this confluence are 3 and 6 respectively, indicating a slightly more rapid decay of the pattern than can be accounted for by simple dilution, particularly for the coarser fraction.

Tidewater extends up the stream channel for a distance of 600 ft from the end of the traverse. Comparison of data on total and exchangeable metals in sediments above and below tidehead shows no significant contrast.

Comparison of Total and Exchangeable Metal in Sediments: Table IV summarizes the data on samples where determinations have been made of both total and exchangeable metal. It is seen that, at least for the Nash Creek samples, a fairly close proportionality exists between exchangeable and total metals, although in the more anomalous samples the ratio of exchangeable to total metal is somewhat higher.

Exchangeable Metal in Sediment Compared with Metal Content of Water: The metal content of stream water was determined at a total of 23 sites, of which six were in drainage from parts of the mineralized area, and 17 were in unmineralized terrain. Measurements were made on the same day to avoid changes due to fluctuations in weather. The data of the six samples from streams draining the mineralized area are given in Table V. In general, it is seen that all anomalous water samples are associated with anomalous sediment samples; beyond this, the correlation is not strikingly close. The median metal content of the other 17 water samples is 0.0006 ppm.

At site 5, water was sampled and analyzed for metals at periodic intervals. Results varied from 0.0017 ppm, when the water level was relatively high, to 0.005 ppm when it was low. This variation suggests that dilution during periods of higher rainfall and runoff can cause a relative decrease in the metal content.

Water Sampling Vs Sediment Sampling in Geochemical Reconnaissance: Both water and sediment sampling have advantages and disadvantages as prospecting tools that should be carefully weighed before being used for a given problem. Both media

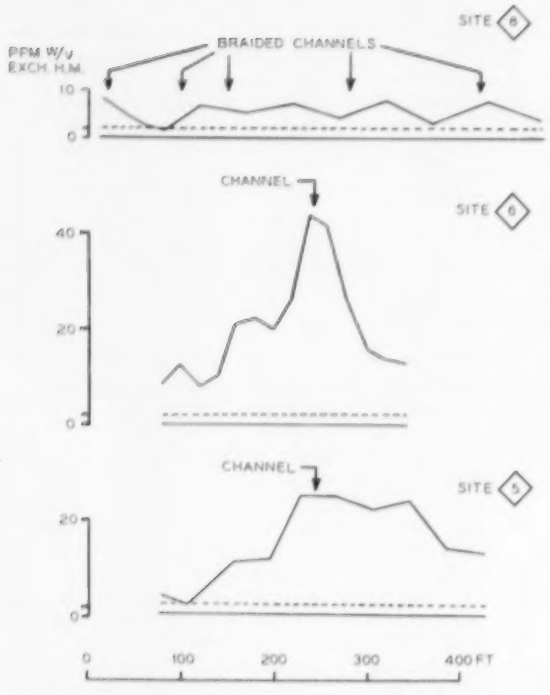

Fig. 4—Exchangeable heavy metal content of sediments from traverses across floodplain.

apparently can be used effectively in the search for geochemical anomalies. The obvious advantage of water sampling is its freedom from sampling errors. In favor of sediment sampling, it should be pointed out that: 1) the metal content is not affected by short-term variations in weather condition; 2) drainage channels may be sampled and tested even though no water is present at the time the samples are taken; 3) a much less sensitive test is required, reducing the necessity for extreme cleanliness of reagents and glassware; 4) small samples are collected, and may be analyzed at a central laboratory with increased precision and efficiency; and 5) the sample may be kept indefinitely for future reference.

Table IV. Ratio of Exchangeable to Total Metal Content of Sediments (—12 Mesh Fraction)

Number of Samples	Range, Exchangeable Heavy Metal Ppm W/V†	Range Total Heavy Metal* Ppm W/W†	Ratio, Exchangeable Heavy Metal : Total Heavy Metal	
			Range	Median
12	1 to 6	80 to 150	0.020 to 0.043	0.031
13	7 to 18	100 to 850	0.011 to 0.072	0.027
8	20 to 45	500 to 1100	0.029 to 0.051	0.041

* Total heavy metal is computed as the zinc content plus one-quarter of the lead content, to allow for the lower sensitivity of the cold-citrate test for exchangeable lead.
† See footnotes for Table II.

In the authors' experience, sediment sampling with analysis by the cold-citrate test is more suitable for geochemical reconnaissance work where large tracts of ground are to be scanned rapidly and cheaply. When exploration based on sediment sampling has led to a strongly anomalous area where the water test need not be particularly sensitive, a detailed survey by water analysis may be more satisfactory.

Applications of Sediment Sampling: During the last three field seasons, the authors took part in a geochemical reconnaissance of an area of 27,000 sq miles in New Brunswick and the Gaspé Peninsula of Quebec. This program involved the collection and analysis of about 15,000 samples of sediment representing more than 4900 sample sites. The reconnaissance work led to the discovery of 43 strongly anomalous areas, of which the nine most promising were designated for intensive physical exploration. To date, four of these nine have been explored by diamond drilling, and in every case have been found to be associated with significant base-metal mineralization.

Table V. Comparison of Exchangeable Heavy Metal Content of Sediments to Heavy Metal Content of Water at Six Anomalous Sites

Heavy Metal Content of Water Ppm	Exchangeable Heavy Metal Content of Sediments — 12 Mesh		— 200 Mesh	
	Active Ppm W/V	Flood-plain Ppm W/V	Active Ppm W/V	Flood-plain Ppm W/V
0.01	110	35	80	70
0.007	140	60	120	120
0.002	90	110	—	—
0.0017	16	20	24	22
0.0017	10	20	14	22
0.0011	18	30	24	35

This experience indicates that the method should be applicable to any lightly glaciated or unglaciated area of temperate climate and well developed drainage. It is probable, therefore, that the method would be effective in the unglaciated areas of the eastern U.S. as well as in New Brunswick. Appraisal of its effectiveness elsewhere must await further experimental work in the field.

Conclusions

Studies of the distribution of heavy metals in sediment and water from streams in the vicinity of a known metalliferous deposit in New Brunswick have shown that:

1) In sediment from the active channel of a stream draining a mineralized area, the content of exchangeable metal varies inversely with grain size but is independent of depth below the stream bed.

2) In sediment from the floodplain of a stream draining a mineralized area, the content of exchangeable metal a) is comparable with that in samples of active sediments of the same grain size taken at the same site, b) varies inversely with distance from the active channel, and c) is independent of depth below surface.

3) The background content of exchangeable metals in both coarse and fine fractions, and in both active and floodplain sediments, is about 2 ppm w/v.

4) Exchangeable metal content of sediments is anomalously high where the metal content of stream water also is high. Analysis of sediments for exchangeable metals, however, has distinct practical advantages over water analysis as a method of reconnaissance mineral exploration in that a) effect of variations in weather is eliminated, b) dry stream channels may be tested, c) the chemical test is

simpler, d) large-scale operations are more efficient, and e) samples may be stored for future reference.

Acknowledgments

The method of geochemical reconnaissance described in this paper is one outgrowth of seven years of intensive research on geochemical methods of exploration in laboratories of the U. S. Geological Survey. It was perfected in the course of exploration work sponsored in 1953 by the Atwater Syndicate of Montreal and in 1954 and 1955 by Selco Exploration Co. Ltd. of Toronto. J. E. Riddell and J. S. Webb, the authors' colleagues in the field during these years, have contributed many ideas that have gone into the development of the method.

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Discussion of this paper sent (2 copies) to AIME before Jan. 31, 1957, will be published in MINING ENGINEERING and in AIME *Transactions*, vol. 208.

Discussion

Energy Transfer by Impact

by J. P. Zannaras

(Mining Engineering, page 47, January 1956; AIME Trans., Vol. 205)

Referring to the article by R. J. Charles and P. L. de Bruyn, let us assume that W = weight of glass bar; P = weight of hammer; e = total deformation; ϵ = unit of deformation; K = potential stress energy; E = modulus of elasticity; L = length of the bar; η = coefficient of inertia; h = height, ft; V = velocity, fps; and a = cross section area.

The portion of kinetic energy which is effective in producing stress energy in a fixed bar struck horizontally is given by the formula[a]

$$K = \frac{1 + 1/3\ W/P}{(1 + \frac{1}{2}\ W/P)^2} P \cdot V^2/2g = Ph$$

where

$$\eta = \frac{1 + 1/3\ W/P}{(1 + \frac{1}{2}\ W/P)^2} \qquad [8]$$

Putting

$$e = W/P = \frac{1 + \frac{e}{3}}{(1 + \frac{e}{2})^2}$$

From the above equation it can be seen that the maximum transfer of kinetic energy to stress energy is when $e = 0$ or $W/P = 0$ which indicates that the weight of the hammer must be very large as compared with the weight of the impacted rod.

Eq. 8 diametrically opposes the conclusions reached by the authors of this article. In fact, if their suggestions were followed to the extreme when $e = \infty$ when $P = 0$, there would be no transfer of kinetic energy to stress energy at all, as η becomes zero.

Eq. 8 presumes that the velocity with which the stress is propagated through the bar is infinite, whereas the authors claim that the compression waves reflected are reaching the struck end of the bar prior to the complete transfer of the kinetic energy to cause such modification of the conditions there as to make them reach the reverse conclusions demonstrated by the above formula.

That such interference exists is unquestionably demonstrated by the authors and others. However, if my observations are correct, such interference for this specific experiment and also for practical comminution is insignificant, and the conclusions of the authors are in error and must be reversed to comply with Eq. 8.

Eq. 5, $w_s = AE\epsilon^2/2$, given by the authors on page 51, is derived from the following equation (Eq. 9): $K = 1/2Pe$, where $P = Sa$, $S = \epsilon E$, $e = \epsilon L$, and $L = 1$.

The above formula, Eq. 9, cannot be applied in this case. This formula is applicable for static loads where the load increases from zero up to its final value, P, in such a way that the deformation at different instants is proportional to the loads acting at those instants and actually represents the *area of a right triangle in the strain load diagram* of base e and height P.

The typical photographs shown in Figs. 3 and 4 represent the familiar strain load diagrams, and since the line of the wave marks the existence and intensity of the strain with the unquestionable conclusion that such strain has been caused by the action of a load acting continuously all along the wave until it reached the horizontal axis, the work stored at this point is represented by the area under the wave line and the horizontal axis and not by the area of the fictitious triangle given by the authors.

Then if this is correct, even visual estimation of these areas at gage stations given in the typical photographs of Figs. 3 and 4 suffice to contradict the authors' calculation given in Figs. 6a and 6b and Figs. 7a and 7b. The typical photographs presented by Charles and de Bruyn show a considerable variation of the intensity of the strain at different stations but very small variation of areas which actually represent the stress energy at the corresponding stations. And, apparently, by squaring the small quantities, the authors magnified their error tenfold.

J. M. Frankland's paper[a] discusses the relative strain intensity and not the total energy for different types of impact loading. He states in his paper, "The reader is explicitly warned not to confuse the results in this report with those obtained when the load is applied by a blow as from a hammer. In this case the peak load rises to very large but mostly unknown values. The accompanying large deflections and stresses are the result of high values of P, not of the dynamic load factor n.

According to Frankland "the dynamic load factor" is the numerical maximum of the response factor. It therefore appears that the authors followed the same procedure in obtaining the relative strain energy ab-

sorbed in Fig. 9 as they did above by squaring the dynamic load factor. This is incorrect if my point is correct that Eq. 9 is not applicable in this case and that the stored energy is represented by the area under the wave line and the horizontal axis and not by the fictitious area of a triangle with base the maximum strain and height the corresponding load.

Fig. 9 is not Frankland's figure. It is the authors' figure obtained from Frankland's data. The correctness of their method is questioned by this discusser.

On page 53 of their article the authors state: "If the interpretation of the experimental results is correct in that light impacting hammers in rebounding cause only slight damping of the vibrations set up during impact then the use of small impacting masses in comminution may have another advantage." This statement clearly shows a misconception of maximum energy transfer by impact, because it is known from elementary mechanics that the kinetic energy is scalar quantity that is independent of direction. All the kinetic energy of the rebounding light hammer is a lost energy.

To prove this mathematically let us take a light hammer of mass M_l and heavy hammer of mass M_h. Assume that the kinetic energy of both hammers is the same. Then if V_l is the velocity of the light hammer and V_h the velocity of the heavy hammer, the kinetic energy of the hammers is

$$\frac{M_l V_l^2}{2} = \frac{M_h V_h^2}{2}. \qquad [10]$$

The velocity of the rebounding light hammer is V_l and the velocity of the heavy hammer in rebounding is $V_h = 0$. The energy transferred to the impacted mass in the case of the light hammer is equal to $M_l \frac{(V^2 - V_l^2)}{2}$ while the energy transferred by the heavy hammer is equal to

$$M_h \frac{(V^2 - 0)}{2}. \qquad [11]$$

From the above equations it can be seen that the heavy hammer transferred more kinetic energy to the impacted mass than the light rebounding hammer.

Another objection to the conclusions of the authors as far as actual comminution is concerned is the form of the impacted mass in a paper, submitted by the present writer for publication, entitled "Mathematical Repeal of the Existing Theories of Comminution. A New Approach to the Principle of Comminution." I have demonstrated by calculus that the determining factor of a solid for storing energy depends to a much greater extent on the form of the solid than on its volume. It appears, therefore, that the capacity of the different forms of rock present in actual comminution to store energy cannot be compared with the capacity of the 10-in. glass rod used by the authors, aside from the fact that each piece of rock will have a different period of vibration.

Irrespective of whether or not the conclusions of the authors are right or wrong, or the reverse formula given by Merriman and other authors is right or wrong, in both cases it is shown or claimed that the energy required for comminution depends on the relationship of the impacting and impacted masses. That is, if we have several steps of impact before the desired size reduction is accomplished, the energy required will depend on the kind of steps we choose to take, and since an infinite combination of impacting and impacted masses can be made the total energy will depend on the sum total of the energies of the specific combinations we chose. It is therefore evident that there will be an infinite number of solutions for the energy required for the same size reduction by impact.

Rittinger's theory is interested only in the initial and final states of the areas of the rock and disregards the intermediate steps, which according to the above statement are very important in determining the energy required for crushing by impact.

Kick's theory is the summation of a series of equal steps of energy and disregards the fact that the energy to break a cube of a certain dimension will depend on what size of mass we choose for impacting it.

Merriman's formula shows that as far as energy is concerned it is more economical to crush by slow crushing than by impact.

R. J. Charles and **P. L. de Bruyn** *(authors' reply)*—The authors appreciate the interest shown in their paper. However, they do not agree with some of the remarks made by Mr. Zannaras. As stated by Roark,[†] the type of equation cited by Mr. Zannaras "is based upon an admittedly false assumption, viz., that the distribution of stress and strain under impact loading is the same as under static loading." In other words, the impact is brought on so slowly that vibrational phenomena play no part in the impact process. Such an assumption limits the applicability of the equation to conditions of impact rarely encountered. In a qualitative manner, however, this equation predicts, and perhaps correctly, that at very long impact times the curve in Fig. 8 of the paper under discussion would rise to high ordinate values. For the more general case where impact times are uncontrollably short and vibrational characteristics must be considered, the present work shows that there is a range of short impact times, brought about by light impacting masses, that leads to momentarily high strain energy absorption by a phenomenon similar to resonance.

The oscillographs given in the paper depict strain vs time at various gage sections and are *not* the familiar strain load diagrams as indicated in Mr. Zannaras' discussion. Integration of the areas under these curves would not give values that could be interpreted as "stored work."

The conditions for the application of Frankland's theory of impact are stated in his original paper as follows: 1) The duration of impact t_i is sufficiently long, say a tenth or more of the fundamental natural period T. 2) The impact is distributed uniformly over the structure. 3) The fundamental mode of the structure is uncoupled with higher modes. It is the opinion of the authors that these conditions have been met satisfactorily in the work under discussion and that no uncertainty was involved in measurement of the peak loads generated by the impacts.

With respect to the effect of multiple blows on an impacted object, as outlined in the last paragraph of the paper under discussion, no reference to energy consideration is made. It is stated that due to residual oscillations left in a body when impacted with light masses a resonance, excited by multiple blows, may be set up which may ultimately lead to fracture.

The actual problem of comminution by impact is complex, for the fracturing and loading processes may occur simultaneously. As an illustration of this complexity, experiments, following the work under discussion, showed that extensive shattering of similar glass specimens could be obtained with energies, at high impact velocity, that were less than *one-third* those required to shatter at low impact velocity. Such results had been expected. On the other hand, an unexpected result did occur in that as the levels of impact energies were raised the amount of size reduction per unit energy accomplished by low velocity impact gradually overtook and surpassed that obtained by high velocity impact. On the basis of the above experimental evidence it does not seem possible to state unambiguously that either a high or a low velocity impact is the more efficient in comminution processes.

References

[6] M. Merriman: *Mechanics of Materials*, 11th ed., p. 332. John Wiley & Sons Inc., New York, 1914.

[†] R. J. Roark: *Formulas for Stress and Strain*. McGraw-Hill Book Co., Inc., New York, 1954.

aime news

Mining Branch Annual Meeting News

The Coal Division plans to present five technical sessions at the Annual Meeting in February. These sessions are listed as follows: Mining and Geology; Mining and Economics; Materials Handling and Safety; Dust Collection and Drying; and Washing. Among the 16 papers listed to date are the following: *Continuous Mining in Pitching Seams* by R. M. Von Storch; *Coal for the Steam Plants of the Southeast* by Ralph E. Kirk; *Developments in Belt Conveyors for Mining* by William Hanson; *Thermal Drying Costs for Fine Coal* by Harry Washburn; and *Coal Flotation with Liquid Hydrocarbons* by S. C. Sun and D. R. Mitchell.

The Minerals Beneficiation Division has scheduled seven sessions for the Annual Meeting. They are: Pyrolysis and Agglomeration; Crushing and Grinding; Mill Design; Symposium on Cyclones for Wet Classification; Solids-Fluids Separation and Solution and Precipitation, Operating Control and Materials Handling, and Concentration. The thirty papers which will be presented include the following: *Third Theory Size Reduction* by F. C. Bond; *Mill Operating Records and Accounts* by N. Herz; *Cyclone Classification in the Iron Ore Industry* by F. D. DeVaney; *Solvent Extraction Processes in Florida* by J. A. Barr, Jr.; *Tailings Disposal at the Silver Bell Unit* by Russell Salter; and *Milling Practice at Lavender Pit Concentrator* by H. K. Martin.

The Industrial Minerals Division will hold seven technical sessions at the Annual Meeting: Chemical Raw Materials; Ceramic Raw Materials; Cement, Lime and Gypsum; Mineral Aggregates and Dimension Stone; Ultra-Lightweight Aggregates; a General Session, and a Joint Session with MBD. Among the thirty papers to be presented are the following: *Eurenite—A New Chemical Raw Material* by Warren A. Wagner; *Nepheline* by H. R. Deeth and H. Maidment; *Highlights of Ideal Cement Co. Plant at Houston, Texas* by T. B. Douglas; *A Rapid Method for Determining the Durability of Ledge Rock* by Joseph E. Gray; *Beneficiation of Refractory Minerals* by Leo L. Gill and Earl Leatham; and *Production of Monazite from Alluvial Concentrates* by J. Hall Carpenter and Robert F. Griffith.

The Mining Subdivision will present three technical sessions at the Annual Meeting, as well as a fourth Joint Session with MGG. They are: Shaft sinking; Southeastern U. S. Mining; and a Canadian Mining Session. The fourteen papers include the following: *Long Raises at Homestake* by C. N. Kravig; *Dredging Operations at Steep Rock Lake* by Philip D. Pearson; *Back-Filling at Noranda* by F. E. Patton; and *Uranium Utilization in Retrospect and Prospect* by R. L. Doan.

Mining Subdivision members are invited to join field trips on Thursday and Friday.

The Geology Subdivision's plans call for three sessions at the Annual Meeting. To date, the following sessions are listed: Latin American; Syngenetic Sedimentary Ore Deposits; and Controversial Sedimentary Ore Deposits. Among the papers to be presented are the following: *Geology and Taconite* by Josiah Royce; *Copper Mineralization in the White Pine Deposit—Michigan* by John R. Rand; *A Proposed Placer Origin for Blind River Uranium Ores* by David S. Robertson and Nelson C. Steenland.

Other Divisions of the Mining Branch have planned the following: Geophysics Subdivision—two sessions; Mineral Economics Division—three sessions; and Mineral Industry Education Division—two sessions. More complete information pertaining to the foregoing Divisions will appear in the December issue of MINING ENGINEERING.

Cuba Chosen for Post Convention Tour

AIME's 1957 Annual Meeting in New Orleans will be followed by a post convention tour to Cuba, including a stay in Havana, and a visit to the Mata Hambre Mine. Departure on a non-stop flight from New Orleans Thursday evening, February 28, begins the tour, which will extend to the following Thursday, March 7.

All those participating will stay at Havana's newest luxury hotel, the Hotel Rosita de Hornedo where accommodations include an air-conditioned bedroom, bath, and private veranda facing the ocean. Two large swimming pools are available.

On Friday afternoon, a sightseeing trip through Havana in private cars is featured, and Saturday's program includes a drive through the surrounding countryside and a visit to one of the perfume factories. Luncheon will be at the Rancho Luna, whose open, palm leaf covered dining room is surrounded by large gardens of sugar cane, coffee, and a variety of vegetables. Guests will be entertained with Cuban music and dances, followed by a cockfight exhibition and a Royal Palm climbing contest.

A scenic motor trip to the lovely Valley of Vinares will start from Havana on Sunday morning, with luncheon at Spa San Vincente. The afternoon will be devoted to pleasure excursions through the incomparable flower gardens of the Valley.

On Monday, a trip to the Mata Hambre Mine is scheduled which features a guided inspection tour and a luncheon given by the management. Tuesday afternoon, participants will begin their trip back to Havana over a different highway.

Wednesday, March 6, the last day of the tour, has been left open for individual sightseeing and shopping. The tour will be terminated on Thursday with the flight back to New Orleans.

Mining Branch Scholarship File

The office of the Mining Branch Secretary has established a file containing information on scholarships, grants-in-aid, fellowships, loan funds, and other forms of student assistance which are available to young men wishing to study in the various fields of the Mining Industry. This file and the listings to be compiled, will be used to assist the many young men who request such information. The AIME National Student Relations Committee headed by Dave Beetem of ALCOA suggests this project may also inspire Mining Branch Members to lead their corporations to establish additional scholarships. We would appreciate your sending any *information* on assistance to college men directly to Arnold Buzzalini, Mining Branch Secretary, 29 W. 39 St., N. Y. C.

Third Annual Rocky Mountain Minerals Conference Scores With Sessions, Social Events and Guests

The Third Rocky Mountain Minerals Conference held again at the Newhouse Hotel in Salt Lake City, September 26-28, was an unqualified success. Sponsored by the AIME Utah Section, the Conference drew 650 registrants and a score of outstanding leaders in the minerals industry. Two and one-half days of technical sessions were followed by half day field trips embracing operations in the Salt Lake Region.

Officers of the conference and of the Utah section get together. Left to right: R. C. Cole, General Conference Chairman; A. J. Thuli, Assistant General Conference Chairman; D. F. McElhattan, section Vice Chairman; Neil Plummer, section Chairman; and Glen A. Burt, section Secretary-Treasurer.

Social Events

Among the highlights was a speech by Governor J. Bracken Lee of Utah, who addressed the welcoming luncheon on Wednesday, September 26. His frank and informal views on trends in national and local government proved stimulating. That evening, registrants had a second opportunity to become better acquainted at the parties sponsored by the mining companies and suppliers. Other social events included the minerals luncheon on Thursday at which John C. Kinnear, Jr., EMD chairman, spoke on the importance of human dignity. He cited examples from his wide range of experience, emphasizing the significance of this point in dealing with superiors and subordinates. Winding up the conference were two social functions enjoyed by all: the cocktail party and informal dinner-dance on Friday evening.

Technical Sessions

The wide range of papers presented morning and afternoon on Wednesday and Thursday, ranged from the problems encountered at the Calera cobalt refinery to the prospects for phosphate mining in Israel. The sessions were well organized and avoided the problem of simultaneous scheduling. The sixteen papers presented covered extractive metallurgy, coal mining, oil and gas resources, management problems, mining methods, and other topics. One of the featured speakers on Thursday was Marling J. Ankeny, recently appointed director of the U. S. Bureau of Mines, who detailed what the agency has done for the metal mines in the 11 Western states.

The Minerals Luncheon on Thursday had John C. Kinnear, Jr., as guest speaker. R. C. Cole, General Conference Chairman is shown speaking.

On Friday morning registrants were off to a fast start at the miners' breakfast which featured a fancy menu to be enjoyed unshaven and in western attire. This was followed by an all-morning symposium, *Loading and Transportation*. Under the guidance of J. M. Ehrhorn, seven authors read papers on open pit and underground operations. A comprehensive picture of the newest methods, practices, and equipment was presented by specialists in the various phases of loading and moving ore. This session was well attended and marked by interesting discussion.

Field Trips

Four field trips left promptly at 1:30 Friday afternoon and visitors had a choice of touring the Calera cobalt production plant, the refinery of the Utah Oil Refining Co., the Western Phosphates plant, or the Vitro Uranium mill. The good turnout was rewarded with hosts who made certain that their guests were well informed and had ample opportunity to view the details of the operation.

For The Ladies

The distaff made merry at a coffee party at the Newhouse Hotel, Wed-

nesday morning, and at the luncheon and fashion show on Thursday at the Salt Lake Country Club, in addition to their active participation at the dinner-dance on Friday. Women's auxiliary events at the conference were under the direction of Mrs. F. V. Richard, chairman, assisted by Mrs. Paul Shields, chairman of the WAAIME Utah Section.

Credit Due

Particular credit should go to the many people who contributed to the smooth running organization of the meeting. Notable work was done by the following chairmen: R. C. Cole, general chairman; A. J. Thuli, assistant general chairman; A. G. Kirkland, program; I. K. Hearn, arrangements; P. H. Ensign, entertainment; and J. A. Burke, registration. The officers of the Utah Section which sponsored the conference, were much in evidence: Neil Plummer, chairman; D. F. McElhattan, vice chairman; and Glen A. Burt, secretary-treasurer.

ABOVE: Miner's Breakfast on Friday carried this invitation, "come unshaven, in western garb."

LEFT: Chairman and speakers at the Symposium on Loading and Transportation included: A. G. Kirkland, Program Chairman; A. J. Thuli, Session Host; J. M. Ehrhorn, Moderator; and the speakers, L. W. Casteel, R. F. Love, R. I. Williams, W. H. Wamsley, A. R. Sims, D. I. Hayes, and Tom Berry (not shown).

FIELD TRIPS: Cobalt refinery of Calera Mining Co. on the shore of Salt Lake was one of the plants visited on Friday afternoon. Left: visitors stream off bus. Center, J. S. Mitchell, plant manager of Calera Mining Co. provides description. Right, refinery building.

INDUSTRIAL MINERALS DIVISION

NEWSLETTER

Dear Members of the
Industrial Minerals Division:

Annual Meeting

Plans are well advanced for the IndMD part of the Annual Meeting in New Orleans next February. Program Chairman Tom Gillingham and his able helpers, the Commodity Committee Chairmen, have organized a technical program of seven sessions, one of which is sponsored jointly with the Minerals Beneficiation Division. The details will be published in the December issue of MINING ENGINEERING. Divisional Chairman Ray Ladoo has arranged for a top-notch speaker for the Divisional Luncheon on Wednesday, February 27th, and the Executive Committee will meet that day at 9:30 am in the Creole Room of the Hotel Roosevelt.

Regional Conference

We hope many of you will be among those present at the Northeastern Mining Branch Regional Conference at Hershey, Pa., November 7-10. In addition to the technical program and field trips, there will be a Mining Branch Council meeting on the 8th, and also a meeting of the IndMD Executive Committee at 11:00 am, Friday, November 9th at the Hotel Hershey.

Divisional Directory

The much-needed and long-awaited Divisional Directory is scheduled to go to press this month, climaxing the arduous labors of many persons, under the leadership of Tom Gillingham, Chairman. The Directory consists of three chapters. The first lists in alphabetical order the names of the 600 or so AIME members who have indicated a primary interest in the IndMD. In the second chapter these names are grouped on a geographic basis by state and city, and in the third chapter they are arranged by fields of interest. To the 600 or so AIME'ers whose primary affiliation is with IndMD, the Directory will be sent free of charge, whereas to others there will be a small charge.

Publications Committee

Your new Publications Committee has been announced as follows: Donald R. Irving, chairman; Q. D. Singewald, H. I. Smith, R. W. Smith, and Paul Tyler. These hard-working and unsung heroes will serve you in that capacity for the year which commenced October 31.

The man who will be your Divisional Chairman next year, Tom Kesler, has had the foresight to round up Commodity Committee Chairmen to serve during his tenure. These chairmen will be listed in our next Newsletter.

We understand that other Mining Branch Divisions may soon follow our lead by placing their Newsletter in MINING ENGINEERING.

Our next Newsletter will appear in the February 1957 issue and we hope to have Tom Kesler, Chairman-elect of IndMD, write it. We feel this will be a good way of bringing his ideas and projects for the coming year to the attention of all who are unable to attend our Annual Meeting in February.

Hope to see you in Hershey.—R.M.G.

Mining Problems Discussed At Meeting in New Mexico

The New Mexico Mining Association and the Southwest International Mining Association held a joint convention in Carlsbad, November 1, 2, and 3. John B. Knaebel of The Anaconda Co. presided and introduced Senator Clinton P. Anderson, the featured speaker. Governor John F. Simms of New Mexico and Mayor Elmer Skinner of Carlsbad welcomed the delegates.

Talks at the first of three technical sessions were focused on the economic and administrative problems of the mining industry, while those at the remaining sessions were concentrated on more specific matters of mineral and metal distribution, and discernible trends in public land policy and mechanization. Among the speakers were Austin Fennessy, International Minerals & Chemical Corp.; Lewis Presnall, Kennecott Copper Corp.; J. L. Kimberley, American Zinc Institute; Miles P. Romney, Utah Mining Assn.; Major General Robert Wood, Fort Bliss, Texas; E. S. Walker, N. M. Land Commissioner; and Roger V. Pierce, consulting engineer.

Field trips were taken to potash mines and to the Carlsbad Caverns. A golf tournament, cocktail parties, and dinner added the proper social balance to the gathering.

AIME Publications Chairmen

The Technical Publications Committee has announced the new chairmen of the AIME Divisional Publications Committees. They are: Jack M. Ehrhorn, Mining Subdivision; S. F. Ravitz, MBD; Sheldon Wimpfen, Coal Div.; Donald R. Irving, IMD; H. R. Gault, Geology Subdivision; F. G. Miller, Mineral Industry Education Div.; and M. E. Peloubet, Mineral Economics Div.

Engineer's Council For Professional Development Meets in Detroit

The 24th Annual Meeting of the Engineer's Council for Professional Development was held on Thursday and Friday, October 25-26, at the Engineering Society of Detroit.

A varied program of technical and social events was arranged under the leadership of ECPD president, Thorndike Saville, who also addressed the group on Thursday morning. His speech was followed by talks on *The First Five Years of Professional Development*, by John J. Cronin, C. J. Freund, and A. C. Monteith. The luncheon speaker was Kenneth McFarland.

In the afternoon, simultaneous panel discussions were held on the above topic; Max W. Johnson led the investigation of orientation, training and appraisal, while K. A. Meade was in charge of the panel on continued education and selected reading and H. P. Seelye directed the discussion on professional identification and integration into the community. At the Annual Dinner that evening in the Statler Hotel, guests heard an address by J. H. Foote.

Friday morning featured open meetings of the following committees: Guidance, Student Development, Ethics, Training, and Recognition. The Ethics Committee tackled interesting if thorny problems. L. F. Grant led the examination of salaries: *Is it ethical for an employer to pay salaries to experienced engineers which are too low compared to wages offered recent engineering graduates?* J. J. Uicker presided over the discussion on change of employment: *In a new job may an engineer use information considered confidential, which he has developed for his former employer?* This topic was followed by the question: *Should a research engineer withhold his results, valuable to the profession, at the insistence of his employer?* Lastly the group, led by J. H. Foote, discussed the ethics of a senior student who has promised to accept a position after graduation, considering another offer.

At the open council meeting in the afternoon reports of the standing and special committees were read.

The ladies too, were kept busy with coffee hours, a visit to the Detroit Art Center, a tour and luncheon at General Motors Technical Center, and a visit to Cranbrook and Northland.

Chemists and Mining Men Convene in San Francisco

Western chemists and mining men gathered at the Fairmont Hotel in San Francisco on November 9 to consider ways to strengthen the bond which exists between them. The all-day conference was sponsored by the San Francisco Chamber of Commerce to stimulate business between the two industries.

During the morning session, participants and interested observers gave their attention to a series of formal addresses including *The Chemical Requirements of the Western Mineral Processing Industry* by Raymond E. Byler, *The Mineral Requirements of the Western Chemical Process 'ndustries* by Fred Lohse, *Mineral Production in the Western States* by Richard M. Stewart, and *Possible Future Developments in Western Mineral Production* by Phil R. Bradley, Jr.

A luncheon meeting followed, featuring a speech on *New Developments in the Chemical Metallurgy of the Glamour Metals* delivered by Peter Colefax, president of American Potash & Chemical Corp., L. A. The afternoon was given over to an inter-industry panel discussion on *Ideas for Future Cooperation Between the Chemical and Mining Industries*, led by S. H. Williston.

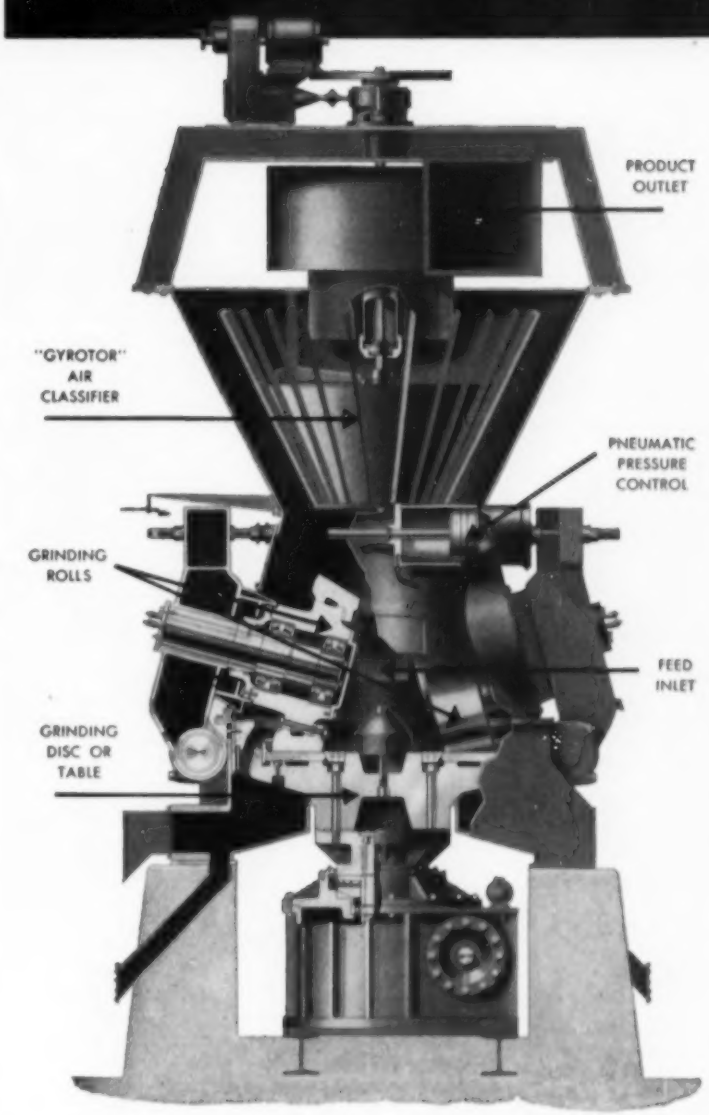
The conference was arranged under the joint chairmanship of Leland A. Doan and Jack How, chairmen, respectively, of the Chamber's Chemical Industries Section and Mining Committee.

NCA Vocational Committee Studies Mining Education

The Vocational Training and Education Committee of the National Coal Association convened at the School of Mines and Metallurgy in Rolla, Missouri, on October 5 and 6. Their conference began in the morning with a brief tour of the campus, devoting special attention to the department of mining building. V. A. C. Gevecker, Assistant Dean, welcomed the participants, discussed enrollment, and gave a statistical report on the school's graduates.

During the afternoon, the chairmen of the various departments discussed their respective curricula as it pertained to mining students. This session was followed by a dinner meeting at which Dean Curtis L. Wilson delivered a brief address on the value of education. Committee members then met with the teaching staff in the department of mining engineering on Saturday morning to go into a more detailed examination of the curricula. During the executive session which followed, it was decided to hold the next conference in May 1957.

DISC-ROLL MILL



The Hardinge Disc-Roll Mill is a roller-type mill with two adjustable, pneumatically loaded rolls for grinding material on a horizontal rotating disc or table (Loesche type). The Hardinge "Gyrotor" Air Classifier, in combination with the mill, provides a complete grinding, classifying, and drying system. A full description is given in Bulletin 52-2.

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Around the Sections

• The **Reno Sub-Section** held their regular luncheon meeting on September 14, in the Nevada Room of the Mapes Hotel. Guest speaker was Governor Charles H. Russell of Nevada, who was introduced by E. L. Stephenson, Section Chairman.

• The **San Francisco Section** met on September 12, at The Engineers Club for their monthly dinner and technical session. Featured speaker for the occasion was Chester Austin, manager of the development section, Steel Plants Div., Kaiser Engineers. Mr. Austin discussed the manufacture of oxygen converter steel, illustrating his talk with a movie on the oxygen converter in operation.

The next meeting of the Section was on Wednesday evening, October 10th at The Engineer's Club, to hear an address by Ian Campbell, Division of Geological Sciences, California Institute of Technology. After cocktails and dinner, the members gave their attention to Dr. Campbell's talk on Petrology and the Industrial Minerals and then watched a 23 min color film entitled *Portrait of the Earth*. In addition to describing the latest developments in airborne photographic and geophysical methods, this film, loaned to the section by Hycon Aerial Surveys, graced the evening with some lovely color shots of California, the Colorado Plateau, and the Peruvian Andes.

• The **Colorado Section** held a dinner meeting on September 19, at the University Club in Golden, with Frank F. Argust, Colorado Fuel & Iron Corp., speaking on *Materials Handling*.

An attendance of over 80 members and their wives attested to the success of the **Adirondack Section's** September 22nd meeting in Fairlee, Vt. The presence of AIME President Carl E. Reistle, Jr., and a field trip to Appalachian Sulfides were the features which attracted this impressive gathering.

Quality Unlimited, **a new 30 min sound-color film on the manufacture of wire rope, has been made available by the Wickwire Spencer Steel Div., of The Colorado Fuel & Iron Corp. The film follows all the operations of wire rope production from basic steel making to finished rope testing, with special emphasis on the physical and chemical tests of quality and on the applications and care of wire rope. For information on obtaining copies, write to** *The Colorado Fuel & Iron Corp., 575 Madison Ave., New York 22, N. Y.*

Utah Section Pioneers New Membership Program

• The **Utah Section** of AIME has pioneered a new idea in its membership development program which has proved outstandingly successful. The plan provides that each member of the section who secures five Utah residents as new members in any one calendar year will receive prepaid dues for the following year.

The plan is completely administered by the local section. It is self-financing, with the section paying dues of those who secure five new members, out of its refund from the AIME New York office of 50 pct of all initiation fees. Since normally 60 pct of the new members join as a Member or Associate Member, the section receives an average of $30 rebate from the initiation fees of five new members.

If you are interested in strengthening your local section, discuss this idea with your officers, and arrange to notify the section membership by letter should you adopt the plan.

The Utah Section's letter to its membership explaining the idea stated:

"The Utah Section of the AIME would like to pay your dues.

"A plan has been evolved whereby any member of the Utah Section who obtains five new Utah Section members in any one calendar year will receive in return prepaid dues for the following year. For 1956 this program will be effective as of April 1, with the termination of the 1956 drive on Dec. 31. Thenceforth each succeeding yearly drive will cover the 12-month calendar period. A large percentage of our present membership should be able to benefit by this offer as there is a very large group of potential AIME members in this area.

"Enclosed are five AIME application forms which upon completion should be returned for further handling. Please be sure to indicate on each application, who is to receive credit for obtaining the new member, as this will be the basis for determining which members qualify for payment of dues.

"There is only one way that a membership committee can do a really outstanding job and that is through the active and enthusiastic participation of the membership as a whole. We have a good product to sell, the Utah Section. Will you help us sell it?"

C. H. Henline
Membership Chairman

Why not start your section's drive next month?

PERSONALS

F. M. CHACE

Frederic M. Chace has been appointed assistant director of exploration, The M. A. Hanna Co., Cleveland. A graduate of Brown University and holder of two advanced degrees from Harvard, Dr. Chace has been a mining geologist in Australia, Peru, Chile, Mexico and Canada.

Walter Granlund has left Kennecott Copper Corp. to join the Empire Zinc Div., New Jersey Zinc Co., Hanover, N. M.

Victor N. Antaki has been transferred from Pocatello, Idaho to the N. Y. offices of Westvaco Mineral Products Div., Food Machinery & Chemical Corp.

Albert L. McGuiness is with Atlas Powder Co., Butte, Montana. He was formerly associated with Frank Stepaneck & Co., Portland, Ore., as estimator-salesman.

J. G. Evans, former manager of Joy Sullivan Machinery Co., S.A., in Mexico, is now at the New York office of Joy Manufacturing Co.

J. M. Van de Plasse is process engineer at United Development Corp., Sydney, Australia.

Carl R. Jokisch, formerly mining engineer, Britannia Corp., Britannia Beach, B. C., Canada, is now at Snow Lake, Manitoba.

Edwin A. Gee, formerly manager of the plants technical section, manufacturing division, E. I. du Pont de Nemours & Co., has been named sales manager of pigment colors. Dr. Gee was assistant chief metallurgist for the U. S. Bureau of Mines before joining Du Pont in 1948.

Alba W. Jeffers, metallurgical engineer, is doing research for Shattuck Denn Mining Corp., Humboldt, Ariz.

F. C. Lendrum, formerly with Ascot Metals Corp. Ltd. is now works manager, Eastern Mining & Smelting Corp. Ltd., Chicoutimi, Que.

Orrin Barrett, formerly with Brazilian National Steel Co., Santa Catarina, Brazil, is now director of coal preparation for The Daniels Co., Indiana, Pa.

Blakemore E. Thomas has left consulting work in Albuquerque, N. M., to join the Geology Dept., San Diego State College, Calif.

John P. Zedalis has resigned from the Army Engineer Corps to work as a field engineer for Charles T. Main Inc., Boston.

John H. Clary has been transferred from Florida to Jamaica, B.W.I., as operations assistant for the Kaiser Bauxite Co.

Arthur E. Adaml, retiring vice president and dean of the college at Montana School of Mines was awarded an honorary Doctor of Engineering degree in recognition of his service to the school and the mineral industry.

Paul L. Goddard, formerly of A. R. Wilfley & Sons Inc., is now associated with Contract Engineering Co., Denver, Colo.

W. A. Coster has joined The American Metal Co. Ltd., N. Y., where he is employed as a geologist.

Emil E. Esckilsen, former consultant at International Talc Co., is now vice president in charge of sales for the Eastern Magnesia Talc Co. Inc.

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We invite you to discuss your drilling problems with a Kennametal representative. He will help you select the right style of Kennametal Rock Bits for each of your operations. Or write for additional information. KENNAMETAL INC., Mining Tool Division, Bedford, Pa.

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- ... sizes up to 5" in diameter
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- ... for use with new high power drills, without adapters

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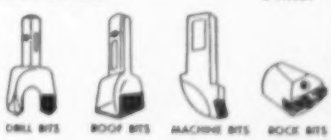

Floyd M. Blanchard has retired from Yuba Mfg. Co., San Francisco.

William H. Bierschenk, formerly with the U. S. Geological Survey, is now geologist-hydrologist, Hanford Atomic Products Operation, General Electric Co., Richland, Wash.

Arthur F. Peterson, Jr., is assistant manager, New York Ore Div., Jones & Laughlin Steel Corp. He had been assistant superintendent, concentrator plant, Bethlehem Cornwall Corp., Lebanon, Pa.

John J. Sheppard, chief engineer, Western Knapp Engineering Co., Hibbing district office, has been named manager of that office. After graduating from the Missouri School of Mines in 1937, Mr. Sheppard engaged in a wide range of mining activities.

Generoso V. Sancianco is chief engineer, Pacific Equipment Corp., Manila, P. I.

G. Y. MURRAY

Gilman Y. Murray is now directing sales activities for the western district of Western Knapp Engineering, a division of Western Machinery Co., San Francisco. After receiving his B.S. and M.S. degrees from MIT, Mr. Murray spent two years as research engineer for the Raw Materials Div., AEC, followed by seven years with Allis Chalmers.

Edward A. Morgan, sub-district supervisor, USBM, is seen presenting the Joseph A. Holmes certificate of honor to Frank Brown, Chino Mines Div., Kennecott Copper Corp., for 47 years' service without a lost time accident. Among the onlookers are Paul Hunter, Chino safety director (3rd from right), and W. H. Goodrich, Chino general manager, (far right).

Arthur O. Hall has left his position as chief geologist, Northwestern Dept., American Smelting & Refining Co. to become exploration engineer for Climax Molybdenum Co., Vancouver, B. C., Canada.

Douglas D. Donald has joined the staff of Behre Dolbear & Co. Inc., mining, geological, and metallurgical consultants, in New York. He was formerly associated with New Jersey Zinc Co., Austinville, Va.

John D. Morgan, Jr., has opened a consulting office in Washington, D. C. Dr. Morgan's recent assignments have been with the Defense Production Administration in Korea and the Office of Defense Mobilization in Washington, D. C.

Thayer Lindsley has retired as president of Falconbridge Nickel Mines, which he founded 28 years ago. He will devote his time to a newly formed exploration company.

P. C. Emrath will be working with the University of Kentucky Contract Team at the University of Indonesia in Bandung.

K. B. Sengupta is senior officer, operations department, Associated Cement Cos. Ltd., Bombay, India.

Charles A. Fellencer has left New York for Bogota, Colombia and a position with GAI Pan American Corp.

John B. Seastone is director, Technical Div., Olin Mathieson Chemical Corp. with offices in New Haven, Conn. He had been research and development manager of the company's metals division.

Paul Kents is resident geologist for New Consolidated Canadian Exploration Ltd., Que. He was formerly senior exploration geologist of Northern Peru Mining Corp.

Robert B. Fulton has left Newmont Mining Corp., N. Y., to begin work with Dawn Mining Co., Spokane.

Henry H. Bruhn, formerly associated with U. S. Potash Co. Inc., is now at U. S. Borax & Chemical Corp., Boron, Calif.

Raymond A. Bradbury, formerly mining engineer with Midwest Utilities Coal Corp., Sparta, Ill., is now in the industrial engineering department of Princess Elkhorn Coal Co., David, Ky.

John J. Spencer, former mining engineer and shift foreman for Castle Dome Copper Co. Inc., is now in the geology department of Miami Copper Co., Miami, Ariz.

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F. W. BOWDISH

Frank W. Bowdish, formerly of the department of mining and metallurgy, University of Kansas, is now at the Institut de Recherches de la Siderurgie, Paris.

Roger O. Oscarson has been appointed area minerals officer of the bureau of land management, with Washington, Oregon, and California included in his territory.

Philip Lindstrom has been transferred from Wallace, Idaho, to Moab, Utah, where he will be in charge of Hecla Mining Co.'s operations.

B. COX

Bart Cox, formerly with Consolidated Ltd., is now an engineer for Canadian Gypsum Co. Ltd., Hagersville, Ont., Canada.

W. S. Hickson has been appointed inspector of mines, Dept. of Mines, Federation of Malaya.

Richard W. Unger is now assistant mill superintendent, Uranium Reduction Co., Moab, Utah.

Ellsworth Y. Dougherty, mining engineer for the SEC in the western states, is now headquartered at 821 Market St., San Francisco.

W. O. Gericke and Associates, of Chicago, Ill., are now available for consultation on coal procurement.

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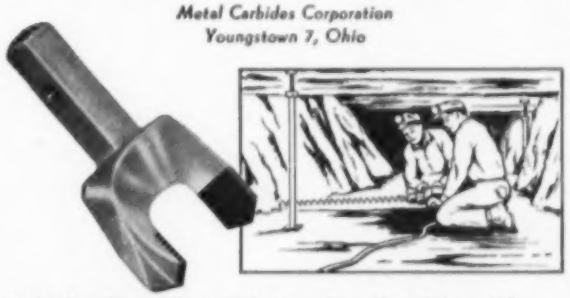

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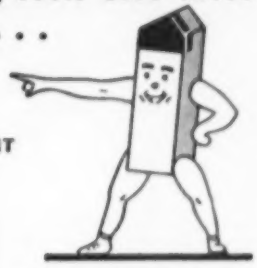

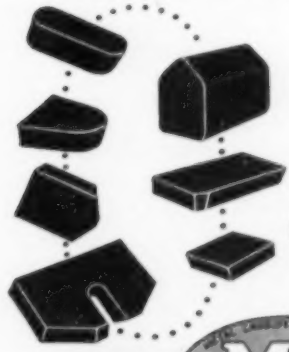

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Herbert R. Hanley has returned to the Missouri School of Mines, Rolla, Mo. where he is Professor Emeritus, to resume limited duties. Since 1954 Dr. Hanley has been academic advisor to Free China's Provincial College of Engineering. He was sent by the U. S. International Cooperation Administration to join the Purdue University team in Taipei, Formosa, under the ICA educational program. In addition to planning and supervising the establishment of a new department of mining and metallurgy, Dr. Hanley lectured, directed laboratory work, conducted student visits to local factories and mines, and personally donated 169 volumes to the departmental library. Professor Hanley has been associated with the Missouri School of Mines since 1923.

R. E. Sinke has been transferred to the Kingston, Jamaica offices of Sprostons Ltd., in the capacity of assistant construction manager.

James A. Kline has left his position as mechanical engineer in the Mine Dept., San Manuel Copper Corp., San Bernardino, Calif. to become field engineer for Kaiser Steel Co., in Fontana, Calif.

Ralph Dunkle has left Homedale, Idaho to take a position with Lepanto Consolidated Mining Co., Mountain Province, P. I.

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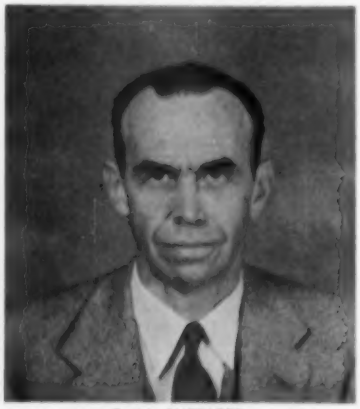
A. E. MOSS

A. E. Moss, chief engineer-geologist of the Iron Ore Co. of Canada, has left Montreal for the company's offices in Schefferville, Que.



P. M. SHERIGER

P. M. Sheriger has been appointed sales manager for metal and nonmetallic mining, Joy Manufacturing Co., Pittsburgh.

Edward M. Laczynski is doing research on rare earth metals at the laboratories of Michigan Chemical Corp., Saint Louis, Michigan.

Max Dessau, Climax Molybdenum Co., has left the Climax offices to become metallurgist in the company's Golden laboratory. **Frank Windolph** will succeed him as mill superintendent in Climax, Colo.

Clyde N. Garman, formerly with Kerr McGee Oil Industries, is now a metallurgist with the AEC in Colo.

A. D. Kranyak is associate petroleum engineer, N. V. Caltex Pacific Petroleum MIJ, Sumatra, Indonesia. He was formerly with The Ohio Oil Co., Hobbs, N. M.

John F. Hunt has left Pacific Northwest Alloys Inc. to become assistant to the president of Carpco Engineering & Manufacturing Co., Jacksonville, Fla.

Carlton D. Weaver, formerly with E. I. du Pont de Nemours & Co., is now sales engineer, Ashland Oil & Refining Co., Ashland, Ky.

John L. Romig manages the quarry and metal mining industry section of the explosive sales department, Atlas Powder Co., Wilmington, Del.

Alfred R. Flinn, resident mining engineer, The New Jersey Zinc Co., has been appointed Eastern manager, with headquarters in New York. Mr. Flinn has been associated with the company for more than 40 years.

The U. S. Chamber of Commerce has announced the following appointments to its 1956-57 Natural Resources Committee: **Frank E. McCaslin,** president, Portland Cement Co.,—chairman; **David L. Francis,** president, Princess Elkhorn Coal Co., **George A. Lamb,** business surveys manager, Pittsburgh Consolidation Coal Co., and **Harry LaViers,** president, South-East Coal Co.,—representatives of the coal industry; **William S. Cumings,** Mining Div., Bethlehem Steel Co., **Endicott R. Lovell,** president, Calumet & Hecla Inc., **Franklin G. Pardee,** president, Lake Superior Iron Ore Assn., **L. J. Randall,** president, Hecla Mining Co., and **O. A. Rockwell,** vice president, The Eagle-Picher Co.—representatives of the minerals and metals industries; **Fred W. Bartlett,** production director, Socony Mobil Oil Co.—representative of the oil and gas producers; and **Harold H. Christy,** power and water department superintendent, The Colorado Fuel & Iron Corp.—representative of water supply, flood control, and reclamation.

C. W. PHILLIPS

Charles W. Phillips is now employed at the scientific laboratory, Ford Motor Co., Dearborn.

Kanji Shiobara, geologist, has been transferred from the Tokyo office of Mitsui Mining & Smelting Co. Ltd. to their Kamioka Mine, Gifu-ken, Japan.

Clifford N. Holmes is director of the strategic minerals section, Phillips Petroleum Co., Bartlesville, Okla.

John B. Myers has been transferred from Libby, Mont., to the Chicago offices of Zonolite Co.

William R. Barton, Jr., U. S. Geolog-

J. T. SHERMAN

John T. Sherman has terminated his association with American Cyanamid Co. to become assistant director for domestic procurement, Raw Materials Div., AEC, Washington, D. C.

Carmelo T. Sison has formed the Republic Resources & Development Corp., of which he is general manager, in Manila. He was previously employed as geologist, Elizalde & Co. Inc., Manila, P. I.

George S. Hume has resigned from the Canadian Department of Mines and Technical Surveys to join Westcoast Transmission Co. Ltd., Calgary, Alberta, as geologist.

Robert M. Dickey, formerly with Bucyrus-Erie Co., South Milwaukee, has joined the staff of American Exploration & Mining Co. in San Francisco.

Kenneth E. Merklin is now at the Hibbing Laboratory of Pickands Mather & Co., Minn. while **F. D. DeVaney** has been transferred from Hibbing to the Duluth office.

F. D. DEVANEY

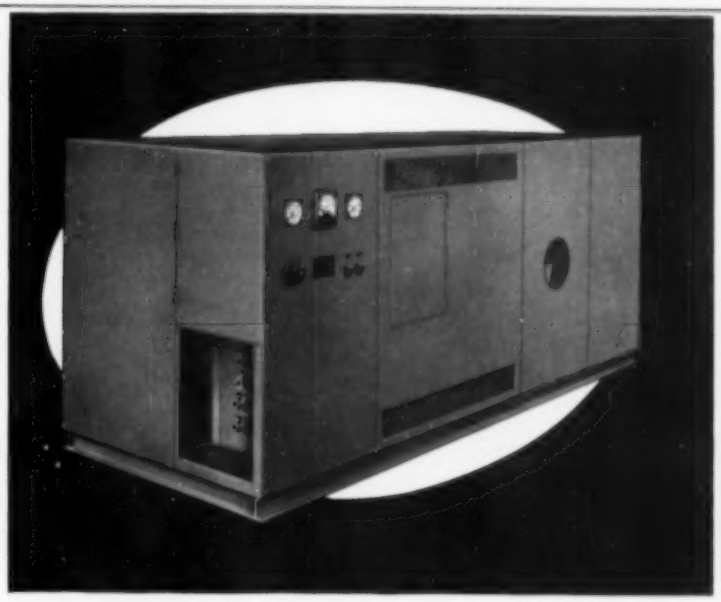
Roland E. Durocher has been named assistant manager, Minnesota Ore Div., Jones & Laughlin Steel Corp. A graduate of Michigan College of Mining and Technology, he joined the company in 1948 as a research engineer.

Ross D. Leisk announced his resignation as vice president of Sunshine Mining Co., Spokane, Wash. He will do a limited amount of consulting work in exploration and the analysis of mining investments. Last year he received an honorary Doctor of Engineering degree from his alma mater, Michigan College of Mining and Technology.

F. G. Woodruff has been named project engineer for the construction of a new smelter at the Ray Mines Div., Kennecott Copper Corp., Ray, Ariz.

Bruce H. Irwin has joined the staff of Stearns-Roger Mfg. Co., Denver.

B. H. IRWIN



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L. KEANE

LeRoy Keane has retired as assistant general manager of Hercules Powder Co., Wilmington, Del., after an association of 38 years.

Ossi E. Palasvirta returned recently from a combined business and pleasure trip to Europe where he visited ore dressing plants in Scandinavia, France, and Germany.

Arthur E. Belton has left Robinson & Robinson Co. to accept a position with Consolidation Coal Co., Fairmont, West Virginia.

W. M. Stoll is now with Macro Div., Kennametal Inc., Port Coquitlam, B. C., Canada.

O. E. Romig, assistant director, Applied Research Lab., U. S. Steel Corp., has been chosen one of three judges of pictorial and technical prints on view at the Photographic Salon, National Chemical Exposition, to be held in Cleveland, November 27-30. Among the judges of pictorial and technical slides is **Walter A. Dean,** works chief metallurgist, Aluminum Co. of America, Cleveland.

John C. Hollister, professor at Colorado School of Mines, has been elected secretary-treasurer of the Society of Exploration Geophysicists for the 1956-57 term.

William R. Barton, Jr., U. S. Geological Survey, has relocated at the Agricultural Research Center, Beltsville, Maryland.

F. Stillman Elfred has been transferred from New York to the East Alton, Ill. offices of Olin Mathieson Chemical Corp.

Gaylord W. Dopson is now concentrator shift boss, San Manuel Copper Corp., San Manuel, Ariz.

Clarence P. Bingham, formerly with U. S. Smelting, Refining & Mining Co., Lark, Utah, is taking a position with Kaiser Aluminum & Chemical Corp., Belem, Brazil.

H. L. Hazen has opened a new company to provide operating management for the uranium mills of his clients in Denver.

H. L. HAZEN

P. D. R. Maltby has joined Con-Met Explorations Ltd., Toronto, as a metallurgist.

Artemios N. Anastassakis has been named inspector of mines for the Ministry of Industry in Athens, Greece.

Robert D. Lowe has obtained an engineering position with United Clay Mines Corp., Hawthorne, Fla.

Henry G. Schuring has left Campbell, Calif. for Atlas Mining & Development Co., Manila, P. I.

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OBITUARIES

Stephen J. Latanision (Member 1951) was killed instantly in a mining accident on July 12th, when a roof collapsed at the Lorado Coal Mining Co., Lorado, W. Va. The 28 year old engineer was a native of Pennsylvania and a graduate of Keystone Jr. College and Lehigh University, where he received the *Old Timers' Gold Watch Award* in 1952. Prior to his college days, Mr. Latanision served in the U. S. Coast Guard for two years. Immediately following the completion of his education, he accepted a trainee position in the production department of Johnstown Coal & Coke Co., Johnstown, Pa. He later became plant engineer at the Crichton No. 4 Mine in Summersville, W. Va. before going on to the Lorado Mines.

Robert C. Pruess (Member 1955) was killed in a plane crash in August, 1956 at the age of 39. A native of Chicago, he had attended Oregon State College and graduated from Colorado School of Mines as a mining engineer in 1942. His first job after college was with Pend Orielle Mines, Metaline Falls, Wash. as junior engineer, followed by a metallurgist assignment with Vanadium Corp. of America in Monticello, Utah. After service as a Second Lieutenant with U.S. Army Ordnance at Aberdeen Proving Grounds in 1943-44, he resumed work with Vanadium Corp., at the Ricran, Peru operations. He later became assistant superintendent of mines, Climax Uranium Co., Colo., after setting up a new operation for Metals & Chemicals Corp. in Costa Rica. Mr. Pruess did consulting work in Florida before becoming manager of Continental Uranium Co., with which he was associated until the time of his death.

Joseph J. Maucini (Member 1949) died last December, at the age of 62. Born in Chicago, he received a B.S. degree in civil engineering from the University of Wisconsin in 1915. After graduation he worked as a civil engineer for the U. S. Bureau of Reclamation, then for the U. S. Bureau of Public Roads as chief location engineer. Mr. Maucini served for a year and a half with the AEF before returning to the Bureau of Public Roads as superintendent of construction. New interests led him to Columbia University for a master's degree in geological engineering, followed by a year of research at the University of Colorado. He joined Marland-Continental Oil Co. as district geologist in 1925, and in 1939 assumed a similar position with Superior Oil Co., Wichita Falls, Texas, where he remained until his death.

Hidenosuke Sano (Member 1936) died some months ago in Tokyo, Japan, at the age of 67. A native of Osaka, Mr. Sano had been associated with the academic life since 1910, when he entered Tokyo Imperial University. He completed his studies there in 1913, and then went on to the Technische Hochschule zu Aachen, Germany, where he remained for two years. After two further years of study at the University of Birmingham, England, Mr. Sano accepted a position as professor of mining at Meiji College of Technology, Tobata, Japan in 1918. He undertook a similar position at Tokyo Imperial University in 1923, where he remained until his retirement. He was named Professor Emeritus there in 1949.

William J. Linney (Member 1935) died of a heart attack at the age of 44 on August 23 at Port Henry, N. Y. A native of Old Forge, Pa., Mr. Linney was a graduate of Worcester Academy and studied at the University of Vermont. He began an eight-year association with Chateaugay Ore & Iron Co. in 1932, working up from assistant mining engineer to superintendent of the mill and sintering plant. In 1940, he joined Republic Steel Corp. as general superintendent of the Chateaugay Div., Adirondack District, and in 1952 was named district manager of Chateaugay and Port Henry.

Joe Crites (Member 1921) died on August 13, 1956. He was 68 years old. A native of Kansas, he studied at Ohio State University obtaining a degree in mining engineering in 1914. After graduation he joined Raymond Bros. Impact Pulverizer Co., Chicago, where he was granted a patent for his invention of a mill for the chemical industry. The year of the stock market crash found Mr. Crites in New York City where he was employed by Combustion Engineering Corp. However, he returned to Raymond Bros. in 1931. His last position was chief engineer for Combustion Superheater Inc., in New York.

Leonard Peckitt (Legion of Honor Member 1883) has died. He was born in Yorkshire, England, in 1860, and came to America in 1882. Immediately following his arrival, he was engaged by the Reading Iron Co., Reading, Pa., as chief chemical analyst in their iron works laboratory. His next position was that of head chemist at the Crane Iron Works, Catasauqua, Pa., where he soon advanced to superintendent, general manager, vice president, and, finally, president. Following the absorption of the Crane Iron Works by the Warren Co., Mr. Peckitt became president of the Warren Foundry & Pipe Corp. He was elected to the AIME Legion of Honor in 1933.

H. Kirk Stephenson (Member 1946) died September 1956. He was program director, earth sciences, of the National Science Foundation, Washington, D. C. Born in Youngstown, Ohio, in 1913, Dr. Stephenson studied at Mt. Union College and received his doctorate from Princeton University in 1941. His varied career included work as field assistant, Newfoundland Geological Survey, and he was later a geophysicist for the USGS in Wyoming, California, and Pennsylvania.

Lyttleton Price (Member 1946) died recently. Born in San Francisco in 1879, he was graduated from the Colorado School of Mines in 1900 with an E.M. degree. His first position was as superintendent and manager at the Minnie Moore Mining Co. in Idaho. In 1909 he became manager-owner of an irrigation project on the Sevier River in Utah, before assuming a managerial post with the Potrero Mining Co., Simaloa, Mexico. Mr. Price set up his own engineering practice in L. A. from 1929 to 1932, and then went back to Mexico with Minas Unidas. He joined the Harvey S. Mudd organizations in 1942, and in 1952 accepted an engineering job with Coronado Copper & Zinc Co., Los Angeles.

Walter B. Longan (Member 1943) has died. He was born in Holden, Mo., in 1885, and attended the Michigan College of Mines. Following his graduation in 1909, Mr. Longan took a job as surface superintendent with Compañia Carbonifera Agujita y Anexas, S. A., in Coahuila, Mex. In 1914, Mr. Longan became general mine foreman of Rosario Mining Co., Honduras, and later spent two years operating and leasing oil fields in Texas and Oklahoma. He entered the field of investment banking in 1922, and in 1933 leased and operated a gold mine in Calif. Subsequently he joined the Bureau of Economic Warfare as an examining mining engineer in Colombia, S. A., and in 1943 started a series of independent mining engineering operations in various cities throughout the West.

Necrology

Date Elected	Name	Date of Death
1942	Charles R. Austin	May 31, 1956
1942	Rex E. Cheek	Unknown
1921	Joe Crites (Senior Member)	Aug. 13, 1956
1947	Adrian Esteve	Oct. 3, 1956
1932	G. R. Gerhard	Unknown
1954	Albert J. Hoch	Feb. 27, 1955
1951	Stephen J. Latanision	July 13, 1956
1935	William J. Linney	Sept. 23, 1956
1943	Walter B. Longan	Unknown
1949	Joseph J. Maucini	Dec. 1955
1924	George C. Newton	May 25, 1956
1883	Leonard Peckitt (Legion of Honor)	Unknown
1946	Lyttleton Price	Unknown
1955	Robert C. Pruess	August 1956
1936	Hidenosuke Sano	Unknown
1946	H. Kirk Stephenson	Sept. 1956
1939	William B. Wells	Mar. 15, 1954
1936	A. S. Wilson	Feb. 26, 1956

MEMBERSHIP

Proposed for Membership
Mining Branch, AIME

Total AIME membership on Sept. 30, 1956, was 25,668; in addition 2,845 Student Associates were enrolled.

ADMISSIONS COMMITTEE

R. B. Caples, Chairman; F. A. Ayer, Vice-Chairman; A. C. Brinker, R. H. Dickson, C. R. Dodson, R. B. Fulton, T. D. Jones, F. W. Hanson, Sidney Rolle, F. T. Sisco, O. B. J. Fraser, F. W. McQuiston, Jr., A. R. Lytle, L. P. Warriner.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

Members

R. L. Broxholme, Tooele, Utah
Richard J. Councill, Wrightsville Beach, N. C.
E. A. D'Acosta, Mexico
John M. Dobson, Stamford, Conn.
Leonard H. Edwards, Orchard Park, N. Y.
Herbert W. Hall, Stamford, Conn.
James M. Jenkins, Cleveland Hts., Ohio

Associates

John N. Butler, Jr., Triumph, Idaho
Clyde Carlisle, Salt Lake City
Lewis P. Evans, Jr., Burlington, Vt.
F. A. Hart, Washington, D. C.
R. L. Mullen, Salt Lake City
Daniel G. Poppe, Moab, Utah
Milne J. Schmid, Rolla, Mo.
Frank Skubic, Virginia, Minn.
Tony Skubic, Virginia, Minn.
W. T. Sparkman, Ajo, Ariz.
Elwin J. Watson, Moss Beach, Calif.
Albert L. Worthen, Jr., New Haven, Conn.

Juniors

E. L. Baker, Birmingham, Ala.
Gustavo Camposano, Colombia, S. A.
Gene E. Congdon, Moab, Utah
Allen H. Ford, Shaker Heights, Ohio
Don Gaines, Portland, Ore.
Thomas M. Hays, Grand Junction, Colo.
Erland Johnson, New Iberia, La.
J. N. Jones, Salt Lake City
E. Kirschner, Mexico
George T. McCrorey, Elvins, Mo.
Wilbur J. Morin, Copperopolis, Calif.
J. B. Morrison, South San Francisco
Tobias R. Peralta, Philippines
Oliver Pierce, Rillito, Ariz.
Wendell L. Reich, Terre Haute, Ind.
J. V. Robles, Oroya, Peru
Earl W. Smith, Grand Junction, Colo.
Howard Tabachnick, Stamford, Conn.
E. P. Terrell, Jr., Avery Island, La.
Erskine Tidmore, Birmingham, Ala.
Donald D. Vincent, Columbus, Ohio
H. J. Vines, Chicago

CHANGE OF STATUS

Associate to Member

Wilford S. Cannon, Spokane
George G. Reed, Jr., Mt. Lakes, N. J.
Donald J. Wyman, Riverside, Calif.

Junior to Member

Peter B. Nalle, Riverside, Calif.

REINSTATEMENT

Members

Robert J. Dalton, San Bernardino, Calif.
Lynn L. McMurray, Spruce Pine, N. C.

Associate

Walter W. Kershaw, Salt Lake City

REINSTATEMENT—CHANGE OF STATUS

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G. M. T. Marshall, Ontario
Robert F. Shurtz, Cleveland

Student to Member

John H. Sullivan, Bishop, Calif.

Junior to Associate

William R. Swiler, Duluth

Student to Associate

Louis M. Lus, Casapalca, Peru

Student to Junior

C. R. Manzano, Chihuahua, Mexico
Thomas H. Riley, Flagstaff, Ariz.

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Coming Events

Nov. 2, AIME, Pittsburgh Local Section, NOHC Pittsburgh Local Section, annual off-the-record meeting, William Penn Hotel, Pittsburgh.

Nov. 2-3, AIME, Central Appalachian Section and Petroleum Subsection, Fall Meeting. (Joint meeting with West Virginia Coal Mining Institute.) Greenbrier, White Sulphur Springs, W. Va.

Nov. 3, AIME, Adirondack Local Section, football game, dinner-dance, Hotel Syracuse, Syracuse, N. Y.

Nov. 5-7, CIM, annual western meeting, Vancouver Hotel, Vancouver.

Nov. 8-10, AIME, Northeastern Mining Branch Conference, Hotel Hershey, Hershey, Pa. Lehigh Valley Section is host.

Nov. 12, 13, Joint Symposium on Mining Research, Missouri School of Mines and USBM, Rolla, Mo.

Nov. 12-15, Amer. Petroleum Inst., annual meeting, Conrad Hilton Hotel, Chicago.

Dec. 5-7, AIME, Electric Furnace Steel Conference, Hotel Morrison, Chicago.

Jan. 9-11, 1957, Fifth Annual Seminar on Spectroscopy and Southern Assn. of Spectographers' Annual Meeting, University of Florida, Gainesville, Fla.

Jan. 17-18, EJC, annual meeting, Statler Hotel, New York.

Feb. 7-9, Colorado Mining Assn., Denver.

Feb. 24-28, AIME Annual Meeting. Roosevelt and Jung Hotels, New Orleans.

Mar. 10-16, EJC Second Nuclear Engineering and Science Congress, Convention Hall, Philadelphia.

Apr. 1-4, Amer. Assn. of Petroleum Geologists, annual meeting, Kiel Auditorium, St. Louis.

Apr. 5, 6, AIME Pacific Southwest Mineral Industry Conference. Sponsored by Nevada Section, cosponsored by San Francisco and Southern California Sections. Reno.

Apr. 8-10, AIME, National Open Hearth Steel and Blast Furnace, Coke Oven, and Raw Materials Conferences, William Penn Hotel, Pittsburgh.

Apr. 11-13, AIME, Pacific Northwest Regional Conference, Portland, Ore.

Apr. 22-24, Second Annual Symposium on Rock Mechanics, Colorado School of Mines, Golden, Colo.

Apr. 22-24, CIM, annual meeting, Ottawa, Ont., Canada.

May 13-16, Coal Convention and Exposition of the American Mining Congress, City Auditorium, Cleveland.

Sept. 8-Oct. 9, Commonwealth Mining and Metallurgical Congress, British Columbia to Nova Scotia, Canada.

Sept. 9-12, American Mining Congress, annual convention, Utah and Newhouse Hotels, Salt Lake City.

Sept. 18-21, International Mineral Dressing Congress, Royal Inst. of Technology, Stockholm, Sweden.

Oct. 9-11, ASME-AIME Coal Div., Joint Solid Fuels Conference, Chateau Frontenac, Quebec.

Oct. 15-18, Northeastern Mining Branch Conference, Hillsboro Hotel, Tampa, Fla.

Oct. 30-Nov. 1, AIME, Rocky Mountain Minerals Conference, Denver.

Nov. 11-14, Society of Exploration Geophysicists, 27th Annual Meeting, Statler-Hilton Hotel, Dallas.

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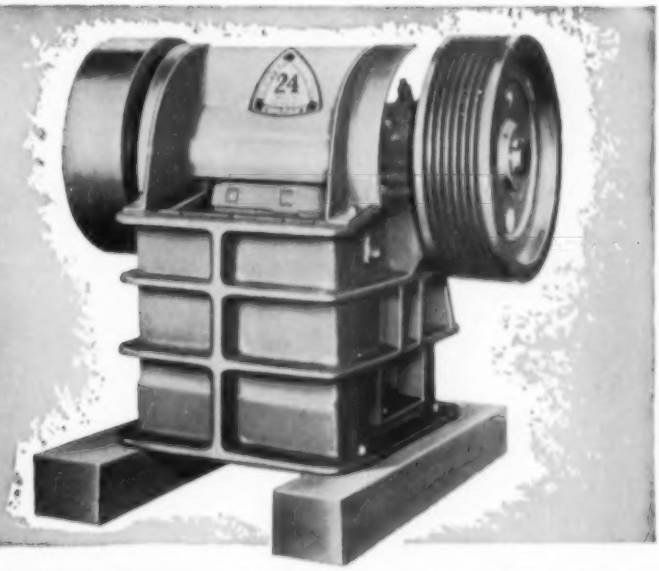

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